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BY

W. L. ZEIGLER

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MILLING METHODS AND COSTS AT THE LEAD CONCENTRATOR OF THE HECLA MINING CO., GEM, IDAHO[1]

By W. L. Zeigler[2]

INTRODUCTION

This paper describing the milling practice of the Hecla Mining Co. is one of a series being prepared by the United States Bureau of Mines.

The concentrator of the Hecla Mining Co., which utilizes combined gravity and flotation methods, is located at Gem, Shoshone County, Idaho, about 2 miles south of the Hecla mine at Burke. It has a maximum capacity of 900 tons per day and all ore treated is drawn from the company's Hecla mine. Milling ore for 1930 contained 8.7 per cent of lead, 1.2 per cent of zinc, and 4.77 ounces of silver per ton.

ORE TREATED

The silver-bearing lead-zinc ores of the Hecla mine occur in a large vein in Burke quartzite. The vein matter, like all veins of the Coeur d'Alene district, is composed principally of crushed and sheared quartzite with some siderite and very small amounts of pyrite. A lamprophyric dyke, ranging from a few inches to 16 feet in width, runs parallel to the vein, sometimes on either side and often in the center.

The economic mineral contained in the ore is chiefly very fine-grained galena which, in size, resembles the grains exposed along a fresh fracture in tool steel. Small amounts of zinc are associated with the galena in the form of dark colored marmatite. The amount of zinc varies greatly in different parts of the mine, but at no place is there any great proportion of it except in the Ore-No-Go vein, which is considered a separate mine. The average grade of milling ore for 1930 had the analysis which follows:

	Per cent
Lead	8.7
Silver, ounces per ton	4.77
Zinc	1.2
Iron	7.1
Insoluble	68.8

The galena occurs in large lenses which run parallel to the strike of the vein; these are located on either side or both sides and at times along the center of the vein. The lens material has the appearance of pure galena but contains from 50 to 53 per cent of lead, 28 ounces of silver per ton, and 3 per cent of zinc. Inclusions of quartz and siderite of

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6600."

2 - One of the consulting engineers, U.S. Bureau of Mines, and mill superintendent, Hecla Mining Co.

microscopic size with some pyrite account for the impurities. On account of the fine-grained character of the galena and associated quartz inclusions, this high-grade ore is harder and more tenacious than the sheared vein material, which makes the ore an ideal one for hand-sorting and jigging operations.

HISTORY

The original concentrator, designed to operate by gravity methods, was constructed in 1888 by the Milwaukee Mining Co. This company at that time operated the Gem of the Mountains mine which was located on the mountain just above the plant. This mill was purchased by the Hecla Mining Co. in 1900 and placed in operation by them two years later.

The original mill had capacity to treat 150 tons of ore per day; the equipment comprised a jaw crusher, rolls, jigs, and buddles. Improved milling machinery was added from time to time as machines were perfected. They were introduced in the following chronological order: The substitution of vanners for buddles, Wilfley tables for sand jigs, fine screens for better sizing operations, slime tables for the old No. 3 Wilfley tables, and finally in 1915, Callow flotation machines were substituted for vanners, filtering equipment was installed to replace settling tanks, and two 6-foot Huntington mills were replaced by a pebble mill.

In 1916 the capacity of the plant was increased to 750 tons per day; a tramway was also installed to carry jig tailing to a gulch located below the mill. At this time a new sorting plant was constructed at Burke with larger crushers and two 36-inch Symons disk crushers for secondary crushing. The structure was of heavy timbers and of ample capacity for the increased concentrator tonnage.

In July, 1923, the mine surface plant, having been completely destroyed by fire, was rebuilt and made fireproof throughout. Steel and concrete ore bins of ample capacity were provided on which the sorting plant was constructed.

During 1925, a flotation unit was completed for the retreatment of jig tailing. This addition made the operation of the tailing tramway unnecessary; it also added materially to the mill extraction and to the profits of the mine. At this time the Callow equipment of the original flotation unit was replaced by a 10-cell 18-inch Minerals Separation Sub A machine; a modern-type hydraulic classifier was also installed to replace a system of fine screens which had been used for sizing table feed.

Early in 1929 all tables were replaced by two Minerals Separation Sub A machines; the latter are used for the selective flotation treatment of material formerly tabled. The ore, as previously noted, contains a small amount of zinc, and a concentration of the zinc content, up to 2.5 per cent, occurs in the fine jig middling. It has been found feasible to produce zinc concentrate from this middling, and the concentrate produced is suitable for shipment to the new electrolytic zinc plant at Kellogg which is partly owned by the Hecla Co.

The lead concentrate and sorted smelting ore are shipped by railroad a distance of 20 miles to the Bunker Hill smelter at Kellogg. Under the present smelting contract it is to the advantage of the company to produce as much product as possible by sorting and jigging. Thorough tests have been made to ascertain the result of treating all the ore by flotation methods, and although higher-grade concentrate could be produced by this plan, it would undoubtedly result in lower extraction and higher milling costs; therefore no net advantage would be gained in view of the present settlement schedule.

PRESENT METHOD OF CONCENTRATING

Crushing and Hand Sorting

A flow sheet of the crushing and hand sorting plant is presented in Figure 1.

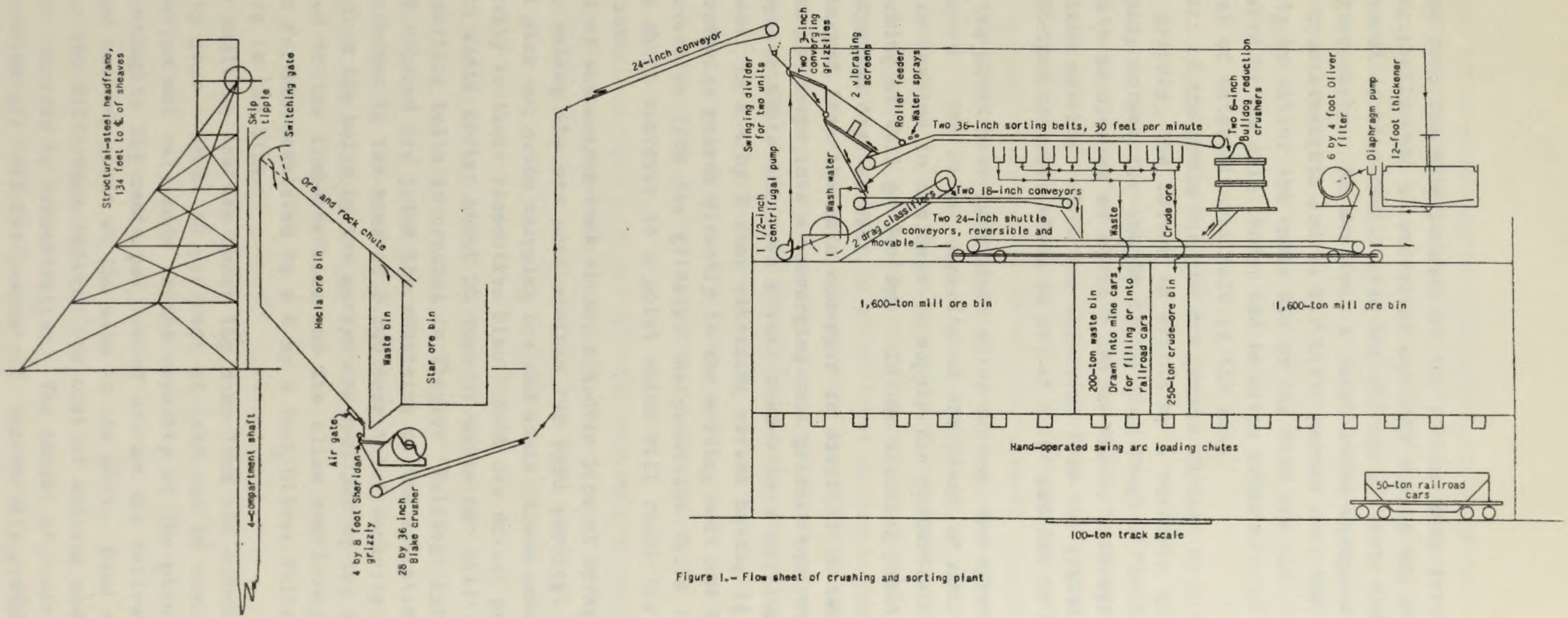

Figure 1.- Flow sheet of crushing and sorting plant

The ore is hoisted from the skip pockets of the various mine levels through a vertical shaft and dumped automatically into a reinforced concrete tipple bin of 2,000 tons capacity. An interesting arrangement of bins facilitates the storage of waste and of ore from the Star and Hecla mines simultaneously. As desired, a steel chute equipped with a swinging-wing apron and actuated by counterweights and an air lift, passes over the Hecla bin and conveys the material by gravity to either the waste bin or the Star ore bin. This mechanism may be quickly raised or lowered by the skip tender and is often moved several times during a shift, depending on the amount of material available in the mine.

From the lower part of the Hecla bin the ore passes through an air-operated gate onto a 4 by 8 foot Sheridan grizzly; the latter is driven by a variable speed motor and acts as both feeder and scalping screen for the 28 by 36 inch Traylor "Bulldog" primary crusher. This crusher is set with an opening of about 5 inches which, although rather coarse, is of decided advantage in hand sorting. Both the undersize from the grizzly and the crushed ore are transported on a 24-inch conveyor which is set at a 16° incline to the top of the sorting plant.

All machines in the plant have individual motor drives, the speed reductions being obtained with gear reducers. The normal capacity of the plant is about 150 tons of ore per hour, and enough ore is handled in six days to supply the concentrator for the entire week when the latter is working at full production. In the crushing plant Sundays are used alternately as repair days and days of rest.

The ore as discharged from the main conveyor is diverted to two separate units by a swinging-wing mechanism and drops onto a converging rail grizzly equipped with 3-inch spaces. The oversize slides down a chute and over a roll feeder to a sorting belt; the undersize passes over a roll feeder to a 4 by 5 foot vibrating screen having $1\frac{3}{4}$ by $2\frac{1}{2}$ inch openings. The vibrating screen oversize passes directly to the sorting belt and forms a cushion on the belt for the larger oversize from the grizzly; the undersize falls into a chute and from there is carried by a short conveyor to a point which will reach the shuttle conveyor, no matter what its position.

Directly in front of the coarse-rock chute, a double line of sprays washes the oversize on the sorting belts, making the ore more visible for hand sorting. Three men placed on each side of each belt pick out crude shipping ore and waste; these products are dropped into chutes and fall by gravity to their respective bins. Each ore sorter produces about 12 tons of crude ore, and each waste sorter about 25 tons of waste per shift. The balance of the material left on the sorting belts is crushed in Traylor "Bulldog" gyratory crushers set at $1\frac{1}{4}$-inch openings. All crushed ore joins the undersize of the vibrating screen on a shuttle conveyor, and is distributed in two bins of 3,200 tons total capacity.

The fines washed from the belts by the sprays are deslimed by two drag-type classifiers; the sands are delivered to the fine-ore bins and the slime overflow pulps to a thickener. The thickened slime is further dewatered by a 6 by 4 foot Oliver filter and the filter cake mixed with the mill ore in the bins.

The sorted crude smelting ore has about the same lead and silver content as the mill concentrate produced by jigs - about 50 per cent of lead and 28 ounces of silver per ton. The hand-sorting operation not only adds to the capacity of the plant but it saves transportation expense amounting to $12\frac{1}{2}$ cents per ton of ore on the railroad haulage of the material from crushing and sorting plant at the mine to the mill. Hand sorting also allows of savings which amount to the difference between the cost of milling and that of sorting, and the losses that would be incurred by concentrating. The amount of crude smelting ore obtained by hand sorting is approximately half the amount of concentrate produced by the mill. All sorted waste is practically clean of mineral and is trammed into the mine for stope filling. No attempt is made to pick either waste or ore of a definite grade; the sorters choose only

waste that shows no mineral and crude ore that is apparently solid galena. The lamprophyric dyke rock which occurs near the ore is very readily visible on the belts and never contains valuable material. On account of the timbering method necessarily used in the mine, large amounts of wooden chips and sticks are sorted from the belts; this in itself would have required a sorting operation before sending the ore to the secondary crushers.

The cost of crushing and conveying ranges from 8 to 10 cents per ton of mine-run ore. The cost of sorting amounts to 20 cents per ton of waste and 35 cents per ton of crude shipping ore.

Crushing and Gravity Concentration

The crushed milling ore is drawn from the storage bins of the coarse crushing and sorting plant into air-actuated bottom-dump steel railroad cars, each of 80 tons capacity; these are hauled over the Union Pacific Railroad, for a distance of 2 miles to the mill at Gem.

A flow sheet of ore treatment at the concentrator is presented in Figure 2.

For reasons previously stated, gravity concentration is carried out as far as possible and is accomplished with Harz-type jigs. The jig middling and tailing are ground and treated by flotation methods in two separate units.

From the mill bin, which is of 800 tons capacity, the ore is fed by 20 plunger feeders, three operating at a time, onto a 24-inch conveyor which discharges the material into a trommel equipped with steel plate screens having 30-millimeter round-hole perforations. A set of 42 by 16 inch Garfield rolls operates in closed circuit with the trommel so that all ore is crushed to pass the screen as undersize before delivery to the main elevator.

The main 14-inch feed elevator delivers the minus 30-millimeter material to two screening units which operate in parallel. Each unit is equipped with trommels which have 18, 12, 7, and 3 millimeter holes. All screens of each unit are intergeared and are driven as a unit at a speed of 18 r.p.m. by a single pulley, which in turn is driven from the line shaft.

The oversize products from the screens are delivered to their respective Harz jigs; the latter operate at various speeds, lengths of stroke, and sieve sizes, depending upon each particular size of feed. The tabulation which follows gives sizes, number of compartments, and operating details of the jigs.

Sizes and operating details of jigs

Size of feed	Number of jigs	Size of jigs: Number of compartments	Size of jigs: Size of sieves, inches	Plunger: Speed, r.p.m.	Plunger: Stroke, inches	Details of sieves used
Minus 30 plus 18 millimer	3	2	20 by 36	150	$1\frac{3}{8}$	$\frac{1}{8}$-inch plate, 6-millimeter holes.
Minus 18 plus 12 millimeter	3	2	20 by 36	165	1	$\frac{1}{8}$-inch plate, $\frac{1}{8}$ by 3/16 inch holes.
Minus 12 plus 7 millimeter	3	2	20 by 36	210	$\frac{1}{2}$	No. 16 brass wire, 6-mesh.
Minus 7 plus 3 millimeter	3	6	20 by 36	215	$\frac{3}{8}$	No. 18 brass wire, 8-mesh.
Minus 3 millimeter plus 16-mesh	3	8	20 by 36	260	5/16	No. 18 brass wire, 6-mesh.

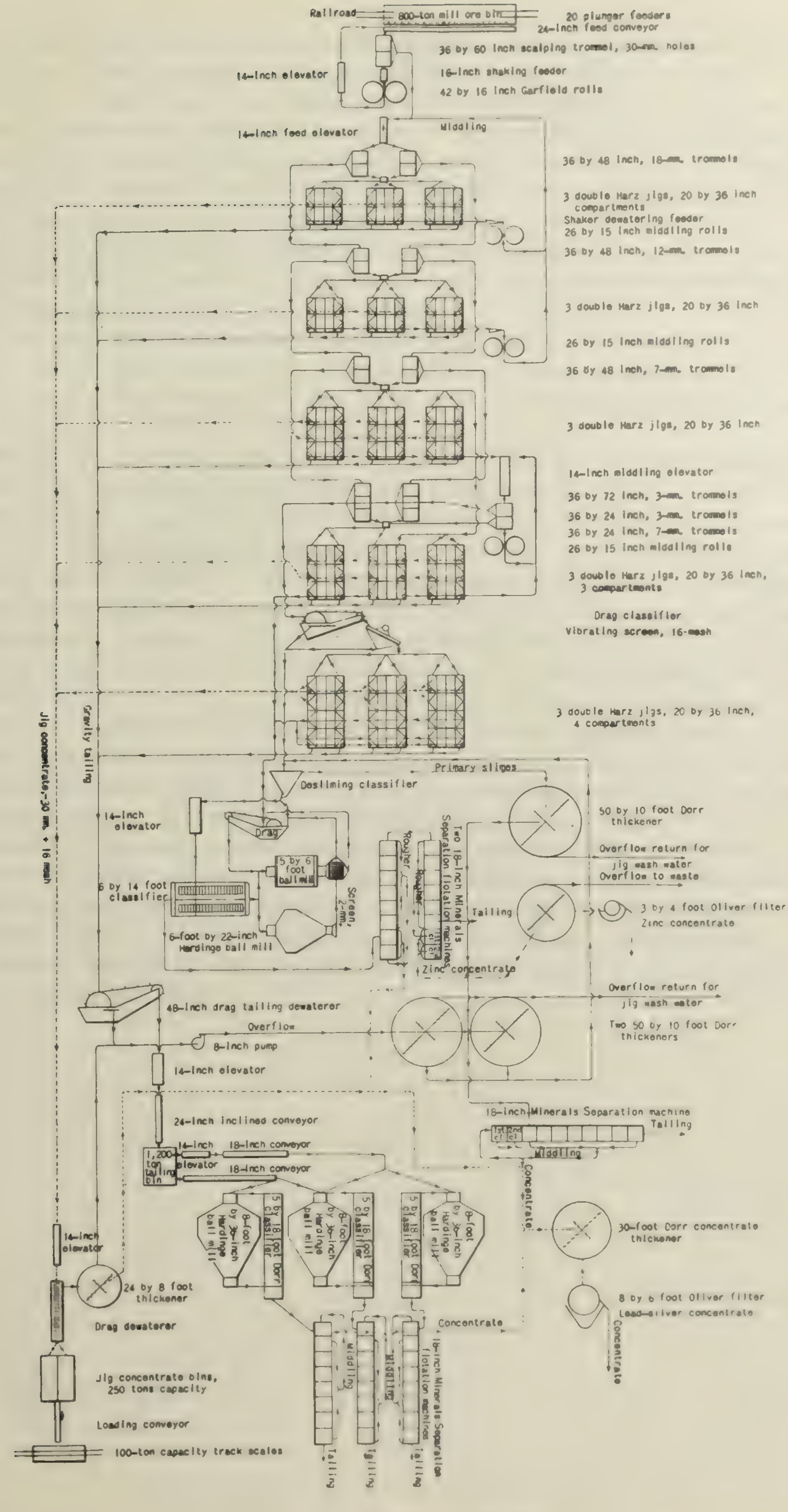

Figure 2.- Flow sheet of ore treatment at the concentrator

Jigs which handle material coarser than 16-mesh size produce finished concentrate, middling, and tailing. The concentrate is produced from the cup discharges of all compartments except the last. Middling is produced from the cup discharge of the last compartment and is crushed by three sets of 26 by 15 inch rolls. One set of rolls serves each of the two coarser jigs, and the remaining set handles the combined middling of the two finer jigs. The crushed middling of the two coarser jigs is returned to the 18-millimeter trommel and those of the two finer jigs to the 3-millimeter trommel.

The sieves of the fine jigs are bedded with 12-millimeter concentrate produced in the coarse jigs, and most of the concentrate produced in the fine jigs is drawn from the hutches after passing through the bed and sieve.

All jig concentrate is conveyed by launders to a central 14-inch elevator which feeds a drag dewaterer. After dewatering, the concentrate is stored in bins of 250 tons combined capacity until shipped. The loading of box cars with concentrate is accomplished by means of portable conveyors.

The jig middling of the fine jigs joins the direct-feed minus 16-mesh sands and is dewatered in preparation for grinding and selective flotation treatment.

The tailings from all jigs are conveyed by launders to a 24-foot by 48-inch dewatering drag classifier. The overflow pulp is joined by the overflow water from the concentrate dewatering classifier and is pumped to two 50 by 10 foot Dorr thickeners. The thickener underflow pulps contain about 10 tons of solids per day in the form of slimes which have been produced by abrasion during jigging. These slimes assay 50 per cent higher in valuable constituents than the original mill feed and are returned to the desliming classifier which handles the original minus 16-mesh mill feed preparatory to flotation treatment. The thickener overflow water is reused at certain times in the year as jig wash water.

As previously mentioned, the ore is ideal material for jigging; very little abrasion of the concentrate particles takes place in the jig beds and practically all concentrate discharged from the cups has perfectly sharp edges and corners.

Table 1 presents a typical screen-assay analysis of the mill feed after crushing to minus 30-millimeter size and indicates the per cent weights of the various sizes of jig feeds and the distribution of the valuable contents in these sizes. Tables 2 and 3 give typical screen-assay analyses of jig concentrate and jig tailing, respectively.

The metallurgical results of the jigging operations, including assays of jig products and recoveries, are given in Table 4.

Table 1.--Typical screen-assay analysis of mill feed

Screen size	Weight, per cent	Assays: Silver, ounces per ton	Assays: Lead, per cent	Assays: Zinc, per cent	Per cent of total metal: Silver	Per cent of total metal: Lead	Per cent of total metal: Zinc	Per cent of total metal through preceding screen size: Silver	Lead	Zinc
Minus 30 plus 18 millimeter	16.2	3.3	7.7	0.8	5.5	6.1	11.2	100.0	100.0	100.0
Minus 18 plus 12 millimeter	28.1	4.5	9.8	1.1	29.7	30.3	26.6	94.5	93.9	88.8
Minus 12 plus 7 millimeter	20.7	4.5	10.1	1.0	22.0	23.1	17.9	64.8	63.6	62.2
Minus 7 plus 3 millimeter	16.4	5.1	10.4	1.3	19.6	18.8	18.4	42.8	40.5	44.3
Minus 3 millimeter plus 16-mesh	7.8	4.7	10.0	1.5	8.6	8.6	10.0	23.2	21.7	25.9
Minus 16 plus 28 mesh	3.3	4.5	8.5	1.8	3.5	3.1	5.5	14.6	13.1	15.9
Minus 28 plus 48 mesh	2.2	4.8	9.5	1.9	2.5	2.3	3.6	11.1	10.0	10.4
Minus 48 plus 65 mesh	1.1	5.8	11.9	1.8	1.4	1.4	1.6	8.6	7.7	6.8
Minus 65 plus 100 mesh	1.1	6.3	12.4	1.6	1.6	1.5	1.5	7.2	6.3	5.2
Minus 100 plus 200 mesh	2.1	7.5	13.8	1.5	3.7	3.2	2.7	5.6	4.8	3.7
Minus 200-mesh	1.0	8.4	14.8	1.2	1.9	1.6	1.0	1.9	1.6	1.0
Composite, calculated	100.0	4.56	9.06	1.16	100.0	100.0	100.0			

Table 2.--Typical screen-assay analysis of combined jig concentrate

Screen size	Weight, per cent	Assays: Silver, ounces per ton	Assays: Lead, per cent	Assays: Zinc, per cent	Assays: Iron, per cent	Per cent of total metal: Silver	Lead	Zinc	Iron
Minus 30 plus 18 millimeter	8.6	27.5	53.6	3.4	8.9	8.7	8.8	6.4	8.1
Minus 18 plus 12 millimeter	23.0	27.9	53.6	3.7	8.8	23.9	23.8	18.6	21.6
Minus 12 plus 7 millimeter	24.7	27.1	52.2	3.9	9.0	25.0	24.9	21.0	23.7
Minus 7 plus 3 millimeter	20.2	28.0	52.1	4.8	9.6	21.0	20.3	21.2	20.7
Minus 3 millimeter plus 16-mesh	12.0	23.2	46.7	6.7	11.1	10.3	10.8	17.6	14.2
Minus 16 plus 28 mesh	5.5	23.3	46.7	7.6	11.3	4.9	5.0	9.2	6.6
Minus 28 plus 48 mesh	2.9	26.4	52.0	5.5	9.1	2.9	2.9	3.5	2.8
Minus 48 plus 65 mesh	1.0	27.4	54.4	4.7	8.4	1.0	1.1	1.1	.9
Minus 65 plus 100 mesh	.8	29.3	57.1	4.1	7.9	.8	.8	.7	.7
Minus 100 plus 200 mesh	.8	32.6	65.3	2.9	5.8	.9	.9	.5	.5
Minus 200-mesh	.5	34.8	71.7	2.0	3.9	.6	.7	.2	.2
Composite, calculated	100.0	26.9	51.9	4.6	9.4	100.0	100.0	100.0	100.0

Table 3.—Typical screen-assay analysis of combined jig tailing

Screen size	Weight, per cent per cent	Assays: Silver per ton ounces	Assays: Lead, per cent	Per cent of total: Silver	Per cent of total: Lead	Per cent of total metal through preceding screen: Silver	Per cent of total metal through preceding screen: Lead
Minus 30 plus 18 millimeter	14.8	0.7	0.9	18.2	16.3	100.0	100.0
Minus 18 plus 12 millimeter	32.0	.5	.7	27.7	27.4	81.8	83.7
Minus 12 plus 7 millimeter	24.6	.5	.7	21.4	21.1	54.1	56.3
Minus 7 plus 3 millimeter....	17.3	.6	1.0	18.1	21.2	32.7	35.2
Minus 3 millimeter plus 16-mesh..................	8.1	.8	1.1	11.3	10.9	14.6	14.0
Minus 16-mesh..............	3.2	.6	.8	3.3	3.1	3.3	3.1
Composite, calculated...........	100.0	0.57	0.82	100.0	100.0	-	-

Table 4.—Concentrator metallurgical results of all units for the year, 1930

	Weight, tons	Assays: Silver per ton ounces	Assays: Lead, per cent	Assays: Zinc, per cent	Per cent of total metal based on original mill feed: Silver	Lead	Zinc	Per cent of metal based on metal fed to each unit: Silver	Lead	Zinc
General mill feed..........	235,413.20	4.77	8.71	1.2	100.0	100.0	100.0	-	-	-
Total lead concentrate	37,218.05	29.01	53.73	4.5	96.1	97.4	65.0	-	-	-
Total zinc concentrate	1,545.15	10.10	9.30	46.7	1.4	0.7	27.7	-	-	-
General mill tailing....	196,650.00	0.14	0.19	0.1	2.5	1.9	7.3	-	-	-
Jigging:										
Feed to jigs.............	191,623	4.37	8.24	0.86	74.5	77.0	63.2	100.0	100.0	100.0
Jig concentrate........	25,952	27.09	52.02	3.5	62.5	65.8	40.1	83.8	85.4	63.4
Fine jig middling	15,064	3.0	6.3	3.0	4.0	4.6	17.3	5.4	6.0	27.3
Jig tailing.................	150,607	0.6	0.9	0.1	8.0	6.6	5.8	10.8	8.6	9.3
Selective flotation:[1]										
Feed to selective flotation.................	58,853	5.7	9.7	2.3	29.8	27.8	49.9	100.0	100.0	100.0
Lead concentrate	8,673	34.6	61.3	5.0	26.9	25.9	16.6	89.5	93.2	33.3
Zinc concentrate........	1,545	10.1	9.3	46.7	1.2	0.7	27.7	4.7	2.6	55.5
Tailing.........................	48,635	0.4	0.5	0.3	1.7	1.2	5.6	5.8	4.2	11.2
Flotation treatment of jig tailing:										
Combined jig tailing	150,607	0.6	0.9	0.1	8.0	6.6	5.8	100.0	100.0	100.0
Concentrate.................	2,592	31.4	46.0	4.0	7.2	6.0	4.0	90.2	90.1	68.8
Tailing........................	148,015	0.06	0.1	Trace	0.8	0.6	1.8	9.8	9.9	31.2

1-Includes flotation of primary slimes, reground jig middlings and original minus 16-mesh ore.

FLOTATION

The concentrator is equipped with three separate flotation units which were installed for the following purposes:

(a) The flotation treatment of primary slime resulting in the production of lead-silver concentrate.

(b) The grinding and selective flotation treatment of minus 16-mesh original mill feed and jig middling produces lead concentrate and zinc concentrate.

(c) A grinding and flotation unit handles jig tailing and produces lead concentrate.

Flotation of primary slime

The minus 3-millimeter pulp upon passing through the last trommel as undersize contains water and solids in the proportion of 40 to 1. The dilution is due to water brought into the circuit with returned middling, wash water used in screening operations, water returned with jig hutch material, and water added for various other reasons. The minus 3-millimeter pulp is partially dewatered in a 16-inch drag classifier; the classifier sands are fed to a 3 by 4 foot Deister vibrating screen equipped with 16-mesh screen cloth. The screen oversize comprises the feed to the fine jigs; the screen undersize joins the drag classifier overflow pulp for desliming in a V-tank. The V-tank overflow pulp is fed to a 50 by 10 foot Dorr thickener by a 4-inch Coeur d'Alene sand pump; the thickener overflow is clear water and the underflow, containing 40 per cent of solid, is delivered to the primary slime flotation unit by a 3-inch Coeur de'Alene sand pump.

The flotation equipment provided for this unit comprises one 10-cell 18-inch Minerals Separation Sub A machine; the first two cells are operated as cleaners and the remaining eight cells as roughers. The feed is introduced into the third cell, and the froth products of the eight rougher cells are returned to the first cell for cleaning. The flotation reagents used consist of 1.0 pound of soda ash, 0.1 pound of 97 per cent cresylic acid, and 0.03 pound of 25 per cent aerofloat per ton of slime treated. Alkalinity of flotation pulp is maintained at an approximate pH value of 9.0.

At times the mill feed contains small amounts of oxidized lead minerals and practically all of these pass directly into the primary slime. There is not enough of this material, however, at any one time to justify the use of special reagents such as are used for the flotation of oxidized lead minerals.

The flotation tailing goes directly to waste; the finished concentrate pulp is fed to a 30 by 10 foot Dorr thickener which handles the finished lead flotation concentrate pulps of all flotation units.

No difficulty is experienced in maintaining satisfactory grade of concentrate in this unit but the production of tailing of low lead content is difficult on account of the large amount of colloidal material present. The flotation feed contains 95 per cent minus 200-mesh material.

Selective Lead-Zinc Flotation Unit

The spigot sands from the V desliming tank, previously mentioned, join the middlings from the fine jigs and the original minus 16-mesh mill feed material for grinding in one 5 by 6 foot Chalmers and Williams and one 6-foot by 22-inch Hardinge ball mill. These grinding mills operate in closed circuit with one 6 by 14 foot drag classifier. The classifier overflow pulp which contains 35 per cent of solid is fed to the lead unit of the selective floation circuit.

The flotation equipment of this circuit consists of two 8-cell Minerals Separation Sub A machines; one produces lead concentrate and the other zinc concentrate. The machines are equipped with 18-inch impellers which are driven in pairs through V-belt drives by 5-hp., 440-volt, 900 r.p.m., Allis-Chalmers Timken-bearing induction motors. The top spindle bearing of each impeller is a self-aligning roller thrust Shafer, and the bottom bearing is a double-row ball, self-aligning SKF. After the bearings are packed with the proper grease they give three months of service without attention.

The flotation feed enters the second cell of the lead unit; the rougher lead concentrates, produced in cells 3 to 8, inclusive, are returned to the first cell for cleaning. Finished lead concentrate is produced from the first roughing cell and from the cleaner cell. The tailing of the lead unit comprises the feed to the zinc unit. The first six cells of the zinc machine produce rougher zinc concentrates which are cleaned in cells 7 and 8. The finished lead concentrate and zinc concentrate are pumped to their respective thickeners by 2-inch Coeur d'Alene sand pumps.

The tabulation which follows gives kinds and amounts of flotation reagents used.

Reagent	Pounds per ton of ore treated
Lead circuit:	
Soda ash	0.1
Zinc sulphate	.3
Cresylic acid	.1
Aerofloat, 25 per cent	.03
Zinc circuit:	
Copper sulphate	.3
Aerofloat, 25 per cent	.05
Barret No. 4 oil	.05

All solid reagents are crushed to minus 4-mesh size and fed into the circuit by cam-type dry feeders. The zinc sulphate and the soda ash are mixed and fed into the grinding circuit. The cresylic acid and the aerofloat are also mixed and fed to the circuit by scraper-type feeders which are driven by $\frac{1}{4}$-hp. motors through double worm-gear reducers. Copper sulphate is fed in the solid state at the end of the lead circuit and, although the unit is not equipped with a surge or conditioning tank, 50 per cent of the zinc is floated from the first cell of the zinc machine.

Table 5 presents assay-screen analyses of feed and final products of the selective flotation unit; the metallurgical results including assays of products and recoveries are given in Table 4.

Table 5.—Typical screen-assay analyses of feed and final products of selective flotation unit

Screen size, mesh	Weight, per cent	Assays: Silver per ton ounces	Assays: Lead, per cent	Assays: Zinc, per cent	Per cent of total metal: Silver	Per cent of total metal: Lead	Per cent of total metal: Zinc
Feed:							
Plus 48	0.3	-	-	-	-	-	-
Plus 65	2.5	-	-	-	-	-	-
Plus 100	5.5	[1]0.8	1.7	0.7	1.5	1.7	2.4
Plus 200	21.0	1.9	3.2	2.0	9.0	8.1	17.6
Minus 200	70.7	5.6	10.6	2.7	89.5	90.2	80.0
Composite[2]	100.0	4.4	8.3	2.4	100.0	100.0	100.0
Lead concentrate:							
Plus 100	0.5	-	-	-	-	-	-
Plus 200	16.0	[3]27.8	46.9	6.5	14.0	12.9	16.1
Minus 200	83.5	33.8	62.7	6.7	86.0	87.1	83.9
Composite[2]	100.0	32.8	60.9	6.6	100.0	100.0	100.0
Zinc concentrate:							
Plus 100	3.4	-	-	-	-	-	-
Plus 200	42.4	[3]5.9	4.7	54.3	43.0	40.6	48.6
Minus 200	54.2	6.6	5.8	48.5	57.0	59.4	51.4
Composite[2]	100.0	6.3	5.3	52.0	100.0	100.0	100.0
Final tailing:							
Plus 48	0.5	-	-	-	-	-	-
Plus 65	5.0	-	-	-	-	-	-
Plus 100	10.0	[1]0.5	0.70	0.20	22.3	19.8	22.3
Plus 200	23.5	0.50	0.70	0.20	33.8	30.0	33.8
Minus 200	61.0	0.25	0.45	0.10	43.9	50.2	43.9
Composite[2]	100.0	0.34	0.54	0.13	100.0	100.0	100.0

1-Includes all sizes plus 100 mesh.
2-Calculated.
3-Includes sizes plus 200 mesh.

Grinding and flotation treatment of jig tailing

The combined jig tailings are dewatered in a 48-inch drag classifier and the sands until recently were conveyed to a 1,200-ton capacity bin by a 14-inch bucket elevator and a 24-inch inclined conveyor. The bin is made of heavy timbers the top of the bin being about 60 feet above ground level.

A revolving screen equipped with ¼-inch openings was recently installed at the upper end of the 24-inch conveyor. The oversize is delivered, by a short 12-inch flanged conveyor, to a 5-inch Newhouse crusher set at 5/16-inch opening on the closed side. The crushed product with the screen undersize drops into the 1,200-ton storage bin.

The Newhouse crusher is supported by three ¾-inch cables from the roof trusses located over the storage bin and 75 feet from the ground level; no vibration is apparent in the structure. The crusher operates 24 hours per day without attention other than an occasional

visit of one of the operators from the gravity concentration unit. It is driven by a 40-hp., 2,200-volt, 870 r.p.m. induction motor; the latter is mounted on a spider of the crusher and actually consumes but 12 hp. for this particular work. It might be considered that this type of crusher is out of place for the crushing of a product, minus 1¼ plus ¾ inch in size, but due to the limited space available and the arrangement of the plant it is the only type of crushing machine that could be hung on a structure of this kind. Although the machine has not operated for a long time, the increase in capacity of the jig-tailing treatment unit well justifies its use. The tabulation which follows gives a screen analysis of the crushed product.

	Weight, per cent
On 12 mm.	24.22
On 7 mm.	33.09
On 3 mm.	19.82
On 16 mesh........	8.87
On 28 mesh........	3.00
On 48 mesh........	2.20
On 65 mesh........	.90
On 100 mesh......	.80
On 200 mesh......	.99
Minus 200 mesh	2.31
Total........	100.00

The screen analysis does not indicate the actual degree of crushing performed in the Newhouse machine, as the crushed material takes the form of thin slabs; oversize particles of a 12-millimeter screen are about 6-millimeter thick.

The storage bin is equipped with a double-A bottom and with plunger feeders arranged in two rows on either side of the bin; the arrangement of feeders allows any two of the three grinding units to be fed by one row of feeders. The feeders deliver the material to 18-inch belt conveyors, which in turn feed the grinding units. Much trouble is experienced with the fine tailing sticking to the conveyor belts; the only apparent solution of this difficulty is a water spray placed underneath the head pulley and followed by a soft rubber scraper.

The grinding and flotation treatment plant is equipped with three units, each of 275-ton daily capacity. Each unit is provided with one 8-foot by 36-inch Hardinge ball mill which is operated in closed circuit with a 5 by 18 foot Dorr classifier. The classifier overflow pulp of each unit containing 42 per cent of solids is fed to one 8-cell 18-inch Minerals Separation Sub A flotation machine.

The ball mills are driven at a speed of 21 r.p.m. by 150-hp. 2,200-volt, 900 r.p.m. General Electric motors through Falk 5½ B herringbone gear reducers. The mills are equipped with cut-steel spur gears having a pitch of 2½ inches, and no pinions have been replaced in six years of operation.

The classifier rakes are operated at a speed of 30 strokes per minute and handle a circulating load amounting to 300 per cent.

The flotation machines are equipped with helical bevel gear drives; this type of gear has proved very satisfactory, as it operates with little noise or vibration and shows no wear after six years of service. The first cell of each machine produces finished lead concentrate; the remaining seven cells produce middling froths which are returned to the first cell. The combined finished concentrates from the three units are pumped to the thickener, which handles all lead flotation concentrate by a 2-inch Coeur d'Alene sand pump. The com-

bined tailings from the three units are discharged into Canyon Creek, which carries a large volume of water throughout the year.

The auxiliary air for all flotation machines is furnished at 3 pounds pressure by two blowers, each of which has a capacity of 1.5 cubic feet of free air per revolution.

All three units of the tailing plant are operated on a 2-shift basis per day. At first, two units were operated continuously, but later it was found that the plant could be stopped and started with little loss of time or minerals, so that a saving of two operators per day was effected.

It is not necessary to grind the tailing very fine in order to obtain satisfactory metallurgical results, as satisfactory recovery and grade of concentrate may be obtained by grinding to 4 per cent plus 48-mesh and 53 per cent minus 200-mesh sizes. This favorable condition naturally results in large capacity, both in the grinding and flotation machines.

The grinding and flotation plant for the treatment of jig tailing was built as a separate unit to the gravity concentrator. It is constructed of heavy concrete foundations, structural-steel frame, and hollow-tile walls, steel sashes, and corrugated asbestos composition roofing. The structure was made fireproof, as it could be readily converted to treat the total mill feed by flotation methods in case the gravity unit was destroyed by fire.

Metallurgical results obtained in this unit including assays of products and recoveries are given in Table 4.

DEWATERING OF FLOTATION CONCENTRATE

The lead concentrate pulps from all flotation units are combined and fed to a 30 by 10 foot Dorr thickener. In order to maintain the density of the discharged pulp uniformly at 60 per cent of solids the thickener underflow product is elevated to a point sufficiently high above the thickener tank so that part of the stream may be returned to the tank by gravity; this method of operation prevents sudden fluctuations in the density of the thickened pulp.

The thickened pulp not returned to the tank is delivered to an 8 by 6 foot Oliver filter equipped with separate drives for drum and agitator. Twill cloth is used on the drum; No. 10 galvanized wire is wound, with 4-inch spacings, over the filter cloth. The filter cake contains about 7.5 per cent of moisture.

The zinc flotation concentrate pulp is thickened in a 30 by 8 foot Dorr thickener; the pulp density of the thickener underflow pulp is maintained uniform, as for the lead filtering circuit. The thickened pulp is further dewatered by a 3 by 4 foot Oliver filter. As the weight of zinc concentrate to be filtered amounts to but 8 tons per day the filter is not run continuously; when the filter is not operating, the thickened pulp is circulated continuously from the elevator to the thickener tank.

Vacuum is maintained at both filters by one 14 by 8 inch Oliver vacuum pump; this pump is driven from the line shaft which serves the filters. Blow air for the filter cloth is furnished at a pressure of 10 pounds per square inch by a 10 by 12 inch Ingersoll Rand compressor.

SAMPLING

A sample of mill feed amounting to 10 per cent of the total mill tonnage is taken from the undersize of the 30-millimeter screen; this sample is further reduced as indicated in the flow sheet (Fig. 3).

This unit, for the sampling of mill feed, is a recent installation; the results obtained from it are but little better than those obtained from the sample formerly cut out hourly and which amounted, in weight, to only a fraction of the one taken at present. The hand sorting of shipping ore causes noticeable variations in the assay values of the mill feed from day to day. The monthly averages of assay values also vary considerably, but yearly averages check quite closely.

Samples of tailing are taken at three places by mechanical cutters, which are checked for accuracy at frequent intervals. The bulk of the plant tailing is rejected from the flotation unit which treats jig tailing, and these tailings are quite uniform in silver, lead, and zinc contents, regardless of the grade of the mill feed.

Hand samples of the concentrate are taken from the end of the loading conveyor as the cars are loaded. The results from these samples check closely those obtained by the smelter, the latter being used for settlements. A representative of the company checks all sampling operations and assays done by the smelter.

WATER SUPPLY

Water for concentrator use is taken mainly from Canyon Creek about ½ mile above the plant; it is conveyed by a 3 by 4 foot wooden flume to a storage tank located 70 feet vertically above the mill. This flume carried enough water to furnish both power and water for the original mill but was supplemented by a steam power plant for use during extreme weather.

Sewage from Burke and from many residences located along the creek above the mill is a great hinderance to flotation operations during periods of low water. A supply of very pure water, amounting to 400 gallons per minute, is conveyed for a distance of 1 mile to the mill from the Black Bear tunnel in an 8-inch wooden pipe line; this water is used at forty-three places in the plant for spray and wash water.

Domestic water, for the plant and for houses located near the plant, is obtained from Bell Gulch, a tributary of Canyon Creek. However, work in the deep mines has nearly drained this supply.

All natural water is very pure and free of dissolved salts; the pH value is practically 7.

METALLURGICAL AND OPERATING RESULTS

Table 4, previously referred to, gives concentrator metallurgical results of all units for 1930.

The production record of the Hecla Co. reported in yearly periods since 1898 is presented in Table 6.

COSTS

On account of the complicated flow sheet of ore treatment in the plant, it is difficult to segregate costs accurately. Table 7 gives the cost sheet for April, 1931, and the figures given are average figures for months in which no extra installation charges were made against operating expenses.

The costs for the unit which treats jig tailing, based on the actual tons of tailing treated, are given in Table 8. These costs are also included in Table 7, but are based on the total tonnage milled.

The cost sheets are kept in monthly, quarterly, and yearly periods. The milling costs per ton of ore treated are also tabulated with the mine production record since 1898 in Table 6.

Table 6.—Production record and milling costs since 1898

	Ore mined and milled, wet weight, tons			Ore and concentrate shipped, dry weight, tons					Metal contents of products shipped, weight			Milling cost per ton of ore treated
	Mined	Milled	Waste	Concentrate: Lead	Concentrate: Zinc	Crude ore	Leasers' ore	Total	Silver, ounces	Lead, tons	Zinc, tons	
1898	1,242	1,218	–	228	–	24	–	252	9,740	144	–	–
1899	9,331	9,331	–	1,799	–	–	–	1,799	67,690	1,085	–	–
1900	31,118	30,835	–	6,010	–	283	–	6,293	225,662	3,620	–	–
1901	12,649	12,545	–	1,716	–	105	–	1,821	54,144	918	–	–
1902	46,525	46 121	–	7,228	–	600	–	7,828	259,394	4,363	–	–
1903	74,077	71,940	–	10,929	–	2,138	–	13,067	408,472	7,002	–	–
1904	106,104	79,475	17,010	12,678	–	1,222	–	13,900	448,740	7,555	–	$0.366
1905	120,600	95,837	19,334	13,329	–	3,100	–	16,429	568,685	9,007	–	.331
1906	126,632	100,631	20,301	14,054	–	2,597	–	16,651	537,342	8,889	–	.458
1907	120,089	95,431	19,252	11,204	–	7,062	–	18,266	538,391	9,382	–	.401
1908	67,941	53,991	10,892	6,518	–	4,211	–	10,729	289,885	5,438	–	.354
1909	126,779	100,748	20,324	10,597	–	7,757	–	18,354	481,476	9,174	–	.317
1910	124,642	99,049	19,982	10,581	–	7,247	–	17,828	492,421	8,781	–	.378
1911	129,443	102,865	20,751	13,151	–	6,059	–	19,210	493,881	9,433	–	.371
1912	138,960	100,760	30,691	12,984	–	7,138	–	20,122	491,745	9,569	–	.375
1913	141,930	99,305	35,926	13,315	–	6,622	–	19,937	507,236	9,416	–	.325
1914	123,857	90,851	25,772	13,015	–	7,037	–	20,052	509,200	9,479	–	.366
1915	146,675	112,646	22,689	15,199	–	11,015	–	26,214	692,444	12,459	–	.357
1916	250,559	184,209	49,889	24,510	–	16,322	–	40,832	1,195,841	20,109	–	.472
1917	374,213	292,726	67,013	30,334	709	14,474	–	45,517	1,366,960	22,182	254	.384
1918	379,703	309,656	54,913	37,260	543	15,091	–	52,894	1,590,062	25,683	217	.413
1919	208,906	175,102	26,352	19,746	–	7,433	–	27,179	795,058	12,363	–	.555
1920	229,893	185,275	33,955	23,518	–	10,682	–	34,200	924,179	16,859	–	.633
1921	267,825	220,051	34,806	26,795	–	13,180	–	39,975	1,060,139	19,569	–	.429
1922	274,581	236,533	24,293	30,759	–	14,118	–	44,877	1,178,124	21,245	–	.437
1923	156,195	136,912	10,950	19,088	–	8,529	–	27,617	774,316	12,888	–	.561
1924	246,209	227,453	7,792	34,225	–	7,590	–	41,815	1,106,398	19,882	–	.523
1925	329,344	271 435	42,096	37,689	–	18,262	2,496	58,447	1,531,793	27,388	–	.536
1926	346,128	277,996	48,353	38,676	–	19,309	2,578	60,563	1,648,680	29,528	–	.692
1927	309,800	247,056	42,234	40,942	–	19,932	1,212	62,086	1,820,110	30,528	5	.686
1928	312,942	256,104	39,067	40,057	–	18,721	536	59,314	1,740,082	29,502	21	.583
1929	310,753	246,123	45,288	37,396	1,727	18,704	903	58,730	1,668,069	29,472	693	.605
1930	310,879	243,626	49,773	37,218	1,543	17,443	774	56 978	1,560,323	28,751	721	.560
Totals	5,956,524	4,613,836	839,698	652,748	4,522	294,007	8,499	959,776	27,036,682	471,663	1,911	–

Table 7.—Concentrator costs for milling 15,222 tons of ore during April, 1931

	Total costs		Cost per ton of ore milled	
Labor:				
Ball mills	$272.91		$0.018	
Conveyors	31.60		.002	
Electric plant	147.19		.009	
Elevators	37.65		.002	
Flotation	300.30		.020	
Jigs	883.53		.058	
Miscellaneous	95.50		.006	
Ore loaders	391.91		.026	
Power transmission	15.09		.001	
Pumps	11.50		.001	
Rolls	28.75		.002	
Supervision	472.27		.031	
Trommels	-		-	
Total		$2,688.20		$0.176
Supplies:				
Ball mills	75.00		0.005	
Conveyors	124.18		.008	
Cresylic acid and aerofloat	72.09		.005	
Electric plant	30.68		.002	
Elevators	84.24		.006	
Filter press	1.44		-	
Flotation supplies	295.23		.019	
Jigs	73.62		.005	
Launder liners	52.19		.003	
Pipe and fittings	0.92		-	
Power transmission	45.00		.003	
Pumps	29.70		.002	
Rolls	98.25		.006	
Steel and iron	-		-	
Trommels	35.32		.002	
Waste and lubricants	69.80		.005	
Miscellaneous	211.20		.014	
Total		1,298.86		0.085
Electric power		1,281.87		0.084
Tailing plant (see separate segregation in Table 8):				
Labor	493.95		0.033	
Supplies	874.21		.057	
Electric power	818.13		.054	
Total		2,186.29		0.144
Assaying:				
Labor	505.08		.033	
Supplies	177.55		.012	
Fuel	18.00		.001	
Total		700.63		0.046
Fuel		223.86		0.015
Totals		8,379.71		0.550

Table 8.--Costs for the tailing plant based on ore actually treated, estimated to be 9,390 tons, for April, 1931

	Total costs		Cost per ton of tailing treated	
Labor:				
Ball mills	$191.03		$0.020	
Conveyors	5.47		.001	
Crusher	56.79		.006	
Flotation	154.38		.016	
Miscellaneous	5.03		.001	
Supervision	81.25		.009	
Total labor		$493.95		$0.053
Supplies:				
Ball mills	175.00		0.019	
Conveyors	11.17		.001	
Cresylic acid and aerofloat	100.07		.011	
Crusher	13.80		.001	
Elevators	-		-	
Flotation supplies	-		-	
Grinding balls	564.17		.060	
Miscellaneous	10.00		.001	
Total supplies		874.21		0.093
Electric power		818.13		0.087
Totals		2,186.29		0.233

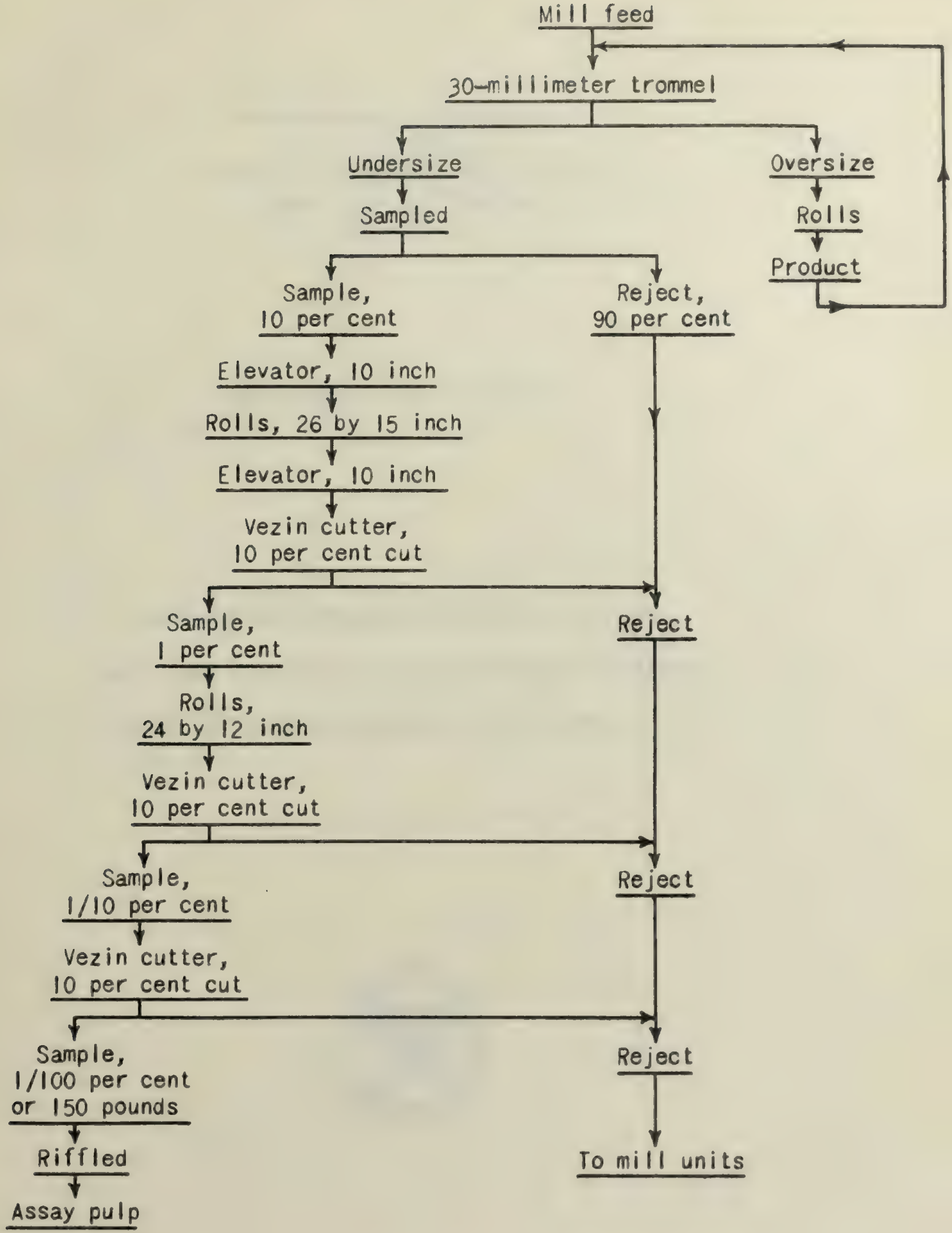

Figure 3.- Flow sheet showing reduction of sample

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE MT. HOPE MINE OF THE WARREN FOUNDRY AND PIPE CORPORATION, MT. HOPE, N. J.

BY

J. R. Sweet

I.C. 6601
April, 1932.

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MINING METHODS AND COST AT THE MT. HOPE MINE OF THE WARREN FOUNDRY AND PIPE CORPORATION, MT. HOPE, N.J.[1]

By J. R. Sweet[2]

INTRODUCTION

This paper describing the mining operations at the Mt. Hope mine, Mt. Hope, N.J., is one of a series being prepared by the United States Bureau of Mines on mining methods, practices, and costs of various mining operations in the United States.

Mt. Hope mine produces magnetite ore. It is situated in the central northern part of the State of New Jersey. The towns of Rockaway, Wharton, and Dover lie within a radius of 4 miles from the mine. Dover is 40 miles from New York City.

During the past five years the mine produced an average of 16,665 tons of ore per month. During 1930, the average monthly production was 14,706 tons, averaging 42.73 per cent Fe content. The ore is concentrated on the property, by dry magnetic separation. During 1930 the average concentration ratio was 1.49, giving a concentrate of average 61.14 per cent Fe content. An average of 85 men were employed underground, and an average of 133.3 men for the mining and milling organization.

The iron concentrate is shipped over the Central Railroad of New Jersey to Wharton, and from there it may continue over the same road or be transshipped to the Delaware Lackawanna & Western Railroad. The greater bulk of the concentrate is shipped to Bethlehem, Pa.

Shrinkage stoping with pillars is the mining method employed. New development is entirely by means of 13 to 15° inclines driven from the 1000 level.

Throughout the report tonnages are given in long tons of 2240 pounds.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6601."

2 One of the consulting engineers, U. S. Bureau of Mines, and chief engineer, Mt. Hope mine, Warren Foundry & Pipe Corporation.

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HISTORY OF MT. HOPE GROUP OF MINES AND EARLY DEVELOPMENTS

The mining of magnetite ore from the Mt. Hope group of mines dates back to pre-Revolutionary days. The oldest workings consisted of what was termed "open work" on the "Jugular Vein," now known as the Taylor ore shoot.

Previous to 1770 the mines had been operated by John Jacob Ford to supply his forges; from 1770 to 1800 they were operated by Jacob Faesch who had leased the property from Ford. Faesch built the Mt. Hope furnace in 1772, and when the Revolutionary War broke out, supplied the American forces with cannon, cannon balls, and other iron equipment.

In 1809, Moses Phillip, jr., purchased the mines and operated them till 1814, at which date the firm of McQuinn and Co. leased the mines. In 1831 the Mt. Hope Mining Co. was incorporated; a few years later the control of this company passed to Edward Bioble, who in 1855 sold the controlling interest to Moses Taylor for $80,000.

With the increased demand for iron ore, the Mt. Hope operations were extended to exploit other ore shoots that outcropped on Mt. Teabo to the southwest, and on Hickory Hill to the northeast; and by 1868 the Mt. Hope Mining Co. was operating nine different mines with an annual production of 72,000 tons.

Previous to 1886 the Scranton Iron Co. produced 6000 tons of ore per month over a period of several years, mining being conducted from a crosscut tunnel 1000 feet in length that encountered five Mt. Hope ore shoots in depth.

In 1870 the Hickory Hill and Mt. Hope mines closed down, but the latter reopened in 1880. The Elizabeth mine at the northeast base of Mt. Teabo was operated continuously throughout this period, however, and produced 30,000 tons annually. Up to 1880 the aggregate tonnage produced from this group of mines is estimated at 1,000,000 tons. Shortly after 1880 the Elizabeth mine closed down, but those on Mt. Hope continued to operate.

In 1882 the New Jersey iron mines first felt the competition of Lake Superior ores, which almost immediately was reflected in a reduced tonnage from this group of mines.

The Brown shaft (the one present operating shaft) was sunk to the 400 level in 1890 and produced continuously till 1893. Mining to the northeast from this shaft encountered the Mt. Hope fault, which displaced the ore beyond the reach of their development work. The shaft was abandoned and allowed to fill with water.

From 1893 to 1898 all of the mines of this group were closed down, the price of iron being down to $2 per ton in 1897. With the improvement of general conditions in 1898, the Elizabeth mine reopened and operated continuously till 1910.

In 1899 the Empire Steel & Iron Co. of New Jersey purchased the Mt. Hope properties, and under its regime the Brown shaft was deepened, the Carlton and Leonard ore shoots were developed by means of shafts through which the ore was mined, and in 1907 a magnetic concentrator was built on the property. In 1910, the aggregate production from this group of mines was estimated at 3,000,000 tons.

In 1922 the Replogle Steel Co. obtained control of the property from the Empire Steel & Iron Co.; from 1924 to 1927 the property was owned by the Warren Foundry & Pipe Co., and operated by Replogle Steel Co. under lease. In 1927 the Warren Foundry & Pipe Corporation was organized and took over the assets of the Replogle Steel Co., and operated the mine under lease to the Warren Foundry & Pipe Co. till December 31, 1930, at which date the Warren Foundry & Pipe Corporation took over the assets of the Warren Foundry & Pipe Co.

The production from 1910 to 1930 was 3,000,000 tons, making a total aggregate tonnage of 6,000,000 tons produced from the Mt. Hope group of mines.

GEOLOGY

The U. S. Geological Survey gives a description of the geology and the mines of this district in Folio 191,[3] and considerable data from that source are embodied in the following discussion.

The Mt. Hope mine is situated in the northeast end of the iron-ore belt that extends northeast from Hacklebarney over a distance of 15 miles to Mt. Hope. Along this zone were a series of almost contiguous iron mines. The ore zone varies in width from 1,000 feet to several miles.

The country rocks are of pre-Cambrian age, the predominant variety being a grey granitoid gneiss. Interlaminated with it are pegmatites and gneisses varying from light colored to dark, well-defined hornblende and biotite phases. Disseminated magnetite is found throughout most of the country rock. The rocks adjacent to the orebodies may be barren of magnetite, or may contain varying amounts of it disseminated throughout the rock. This magnetite is considered as having been formed from the magma that yielded the gneiss. Pegmatite

3 Bayley, W. S., Salisbury, R. D., and Kummel, H. B., Raritan, N.J.; U. S. Geol. Survey Folio 191, 1914, 32 pp., 5 maps.

commonly forms the bottom rock of the oreshoots, and in the pegmatite are often found stringers and patches of magnetite. The pegmatites are considered to be of aqueo-igneous origin, and to have been derived from the same magma that had previously furnished the granitoid gneisses of the region.

Analogous in mode of formation to the pegmatites, the magnetite orebodies were probably formed by true injections and by solutions and gasses from the deep magma.

Orebodies

The ore zone at Mt. Hope is 1,000 feet wide. Within this zone are three main parallel veins striking northeast, equally spaced 500 feet apart, and dipping from vertical to 60° to the southeast. (See fig. 1.)

The oreshoots are tabular shaped, disposed edgewise and when in the same vein lie one above the other. They are parallel in their northeast strike, and are conformable with the structure of the intervening gneiss. The orebodies all dip to the southeast and pitch or rake 14° northeastward.

The four main oreshoots lie within the three veins. Two occur in the central vein, separated vertically by 230 feet of barren gneiss. Three other oreshoots also apparently lie in the central vein (two as projected northeast from the Richard mine and the third, the Spencer, outcropping northeast of Mt. Hope); the vertical interval from the lower of the two Richard mine oreshoots to the cap rock of the Spencer orebody is 3,000 feet.

Although the orebodies exhibit remarkable uniformity in their general attitude, they show local irregularities, mainly in the form of rolls in the foot and hanging walls and pinching and widening of the vein. They vary in height from 150 to 400 feet, and in thickness from a few feet to 40 feet; the average width of the largest deposit being mined is 19 feet. The three present producing orebodies have been developed for 7,500 feet as measured on their pitch downward from the surface.

Faults

The Mt. Hope group of orebodies is traversed by the Mt. Hope fault, which strikes N.78°W. and dips 63° to the southwest. The fault zone is 20 feet wide on the 1000 level. It is a normal fault, the vertical and horizontal displacements being respectively 380 feet and 150 feet. This displacement of the ore precluded continuous operation along the full pitch of the orebodies, and it was necessary to develop them on opposite sides of the fault as separate units, designated as North and South.

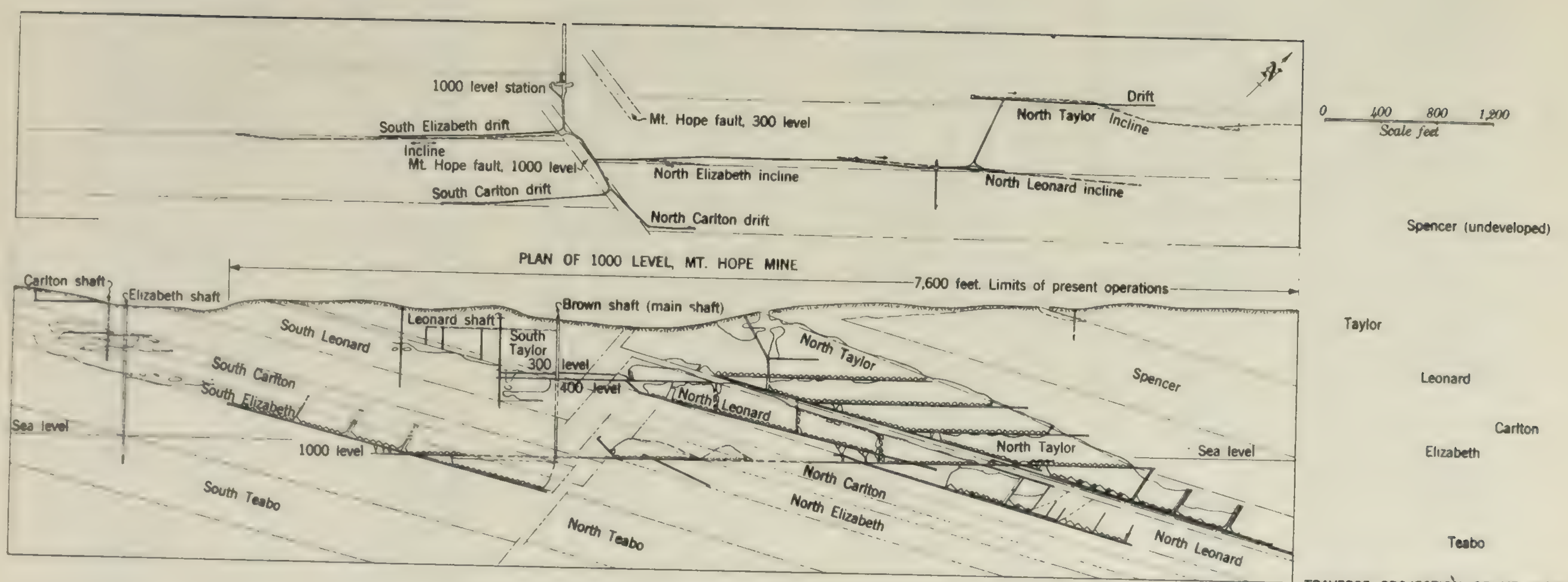

Figure 1. —Longitudinal elevation, Mt. Hope mine

Two of the orebodies have been cut by strike faults. In both instances the displacement has been nearly horizontal, about 300 feet northeastward, and nearly paralleling the strike of the oreshoots. One fault effected a 300-foot separation of the ore along its pitch; it traverses the ore from the hanging to the foot wall and forms the hanging wall of the stope in this section. There is considerable fault breccia composed of ore and rock which readily sloughs from the fault wall, and the movement has badly fractured the adjacent ore.

The second strike fault encountered an orebody near its mid-section and displaced the ore above the fault and below the cap rock for a distance of 300 feet northeastward. The displaced segment was mined as a separate orebody. The fault apparently passes into the cap rock, due to the downward pitch of ore.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCK

Two general types of magnetite ore are found; one, a hard, massive, granular type, and the other a hard, massive, laminated type. The gangue minerals are hornblende, pyroxene, apatite, biotite, quartz, feldspar, and small amounts of pyrite and calcite. The iron content of the ore varies from 35 to 65 per cent. Although both types of ore are found in the various oreshoots, the granular type is mainly limited to the largest, or Taylor ore body. It is hard and tough, breaks large, and will stand unsupported over at least 40-foot widths.

The laminated type is the weaker ore. Its structure is generally normal to the vein walls, and it varies in dip from the horizontal to the vertical, often paralleling the pitch of the ore shoot. It is an easier ore to drill and to break, and breaks smaller than the granular type.

Cross fractures, slips, and faults of small displacement traverse the ore, forming lines of weakness in the backs of the stopes as the ore is mined upward. Their extension into the hanging wall is probably the cause of large blocks of wall rock sloughing off into the stope upon the final withdrawal of the ore.

Lenses and parallel bands of gneiss and of "vein matter" (a micaceous hornblendic schistose rock) which are practically barren of magnetite occur within the ore and lying on the vein walls. The lines of contact of this waste with the clean ore and with the hanging and foot walls form lines of weakness.

The lateral limits of the ore are usually well defined. The ore limit may be in the form of a well-defined wall or it may follow the parallel banding of the gneiss with no other apparent structural line, or it may be limited by the parallel bands of "vein matter" that are found lying on the walls. The bands of "vein matter" vary in thickness from a few inches to several feet; if this waste is not broken with the ore as stoping progresses, it will tend to slough off later when the lateral support furnished by the broken ore is withdrawn.

The top of an orebody may terminate abruptly or gradually. Horses of waste may extend down into the ore and nearly fill the vein walls, causing the ore to pinch to narrow widths, which continue upward following one or both of the vein walls into the cap rock. Near the bottom the oreshoots frequently attain their greatest width; sometimes they pinch out gradually, but often the change is abrupt, as shown in the transverse section of Figure 5. The ore may pinch out above or extend below the average pitch line of the ore.

The wall rock is a hard, solid granitoid gneiss that stands well. All of the oreshoots dip to the southeast at an average of from 60 to 85°, with local irregularities giving dips varying from 45° to the vertical. The axis of the rolls that occur in the walls of the ore may lie in any direction; the larger rolls at times will reverse the general dip of the walls, causing the ore to become heavy on the overhanging footwall. In the central part of the stopes where the flatter dips of the hanging wall are present, there is some slabbing off of the hanging wall and fracturing of the ore as stoping advances upwards and the stope takes weight.

The orebodies attained their greatest height near the surface, 430 feet being the maximum height to which any oreshoot was mined. In the Taylor ore shoot, at 7,000 feet of depth as measured on the pitch, the minable height has decreased to 200 feet, but this loss has been partly compensated by increased widths. The lower stope developed in the Taylor orebody has a maximum width of 35 feet and an estimated average width of 19 feet, and will be mined to a height of 200 feet.

METHODS OF PROSPECTING AND EXPLORATION

The early prospecting and exploration work was done by means of magnetic surveys, diamond drilling, test pits, adit drifts and crosscuts, and shallow shafts. As subsequent mining of the ore revealed its persistence in depth a few of the principal shafts were deepened. From these shafts the oreshoots were explored by means of drifts, crosscuts, and diamond drilling.

The recent prospecting and exploration work has been done mainly by diamond drilling. In a few instances where the continuity of the ore is broken on its pitch by strike faults, prospect raises are driven from the respective inclines to locate the faulted ore.

During 1930, 5,700 feet of diamond drilling was completed. The drilling was done from old openings in the hanging wall of the oreshoots and from the parallel development drifts and inclines of adjacent oreshoots. The drilling was contracted at $3.90 per foot; this price included supervision, supplies, and labor. Records of carbon consumption are not available, but it was probably not much less than 1 carat per 100 feet of hole. The core diameter was 1 3/32 inch. The rock in general was a hard, medium-grained granitoid gneiss. The average drilling speed for the last hole drilled was 11.2 feet per 8-hour shift. The hole was 700 feet deep and pitched downward 71°. Two drilling crews were employed under the supervision of the diamond setter, each crew consisting of a runner and a helper.

Sludge samples were not taken. For core samples, the core was divided visually into sample sections of nearly equal iron content, no sample section representing more than 5 feet of core. This core was then split, one half being assayed and the other half filed in the core boxes. Where the core indicated a mineralized zone which was not of commercial size and grade the complete core was filed; where the core was barren rock, 1 inch specimens from each foot of core were taken and filed, the balance of the core being discarded.

To test for the deviation of the hole in a vertical plane, gelatin was first used. The results obtained were not accurate and its use was abandoned in favor of hydrofluoric acid. The test bottles used were 4 inches long and 7/8 inch in diameter. The test-bottle container was lowered for the first 100 feet by a string of drill rods and to deeper depths by means of a copper wire wound from a reel.

The correction angles to apply to the etched readings were determined by office tests. First, sufficient dilute acid was made up to care for all of the office and drill-hole tests. The strength of the acid was adjusted so that nine hours were required to give a good etching, this being the period that the tests remained in the hole. The test bottles containing the acid were set at the desired angle of the hole to be drilled. The difference between the etched angle and the set angle gave the correction angle for that particular angle and acid.

No goniometer being available, the angle etched on the glass bottle was obtained by first tracing the etched line on the outside of the bottle with black ink, then bringing this line into a vertical plane and making the horizontal trace of the plane coincide with a straight line drawn on a sheet of paper. A line parallel to the edge of the bottle was transformed to the same sheet of paper by means of a straight edge. From these two lines the etched angle was read. The results as indicated by duplicate tests are within the limits of accuracy obtainable in setting a desired angle on a drill machine and in collaring the drill hole.

All of the drill holes flattened as they advanced. Five different holes, angled downward from 60 to 84°, showed deviations at 600-foot depths of from 25 to 30 feet, as measured in the vertical plane and from the projected line on which the hole was started. Following are correction angles that were applied to the etched angles on the test bottles; the readings to minutes are not direct readings, but averages of a series of readings:

True angle		Etched angle		Correction	
Degrees,	minutes	Degrees,	minutes	Degrees,	minutes
18	10	23	05	- 5	22
60	50	66	57	- 6	06
71	25	76	52	- 5	30
76	00	80	23	- 4	23
84	20	86	20	- 2	00
84	40	86	35	- 1	55

METHODS OF SAMPLING AND ESTIMATION OF TONNAGE AND VALUES

Sampling

No samples of the ore mined are taken underground, since visual inspection readily distinguishes ore from waste. On the surface the ore is weighed in 50-ton cars as it is being transported from the mine-shaft ore bin to the mill. In the mill, samples of the ore are taken from a conveyor belt. From these weights and samples the daily tonnage and the average iron content of the ore are obtained.

Where the drifts are in ore, channel back samples are cut at 10-foot intervals. If the ore extends outside of the sides of the drift, test holes are drilled with Leyner machines and the drill cuttings are caught in a powder box. Composite sample-assay and geologic maps are made from which sections normal to the pitch of the oreshoot are constructed.

Estimation of Tonnages and Values

Ore-reserve estimates are made annually, and special estimates are made during the course of the year to check the current extraction with the past estimate. Due to the uniform grade of the ore, tonnage alone is considered in the estimate. The ore-reserve estimate consists of broken ore, assured ore, probable ore, and possible ore.

The broken-ore reserve is obtained directly from the monthly reports and is the difference between the ore broken and the ore extracted. The ore broken is based upon monthly transit surveys of the stopes. At the end of the year 1930, the broken-ore reserve was 63 per cent of the past year's production.

To estimate the ore in place the tons of ore per linear foot of ore body, as measured on its pitch, are estimated. This unit times the pitch distance of the section of oreshoot under consideration gives the ore tonnage in that section. The tons of ore in 1 linear foot of the orebody are sometimes calculated from various sections of the ore revealed by drifts, raises, cross-cuts, and by diamond-drill holes, projection parallel to the pitch of the ore being used to construct cross sections of the oreshoot normal to its pitch. More commonly, however, the tonnage per linear foot of oreshoot is obtained from past records of the tonnage mined from other sections of the same shoot.

Assured ore is the ore in place that has been reasonably proved by diamond drill holes or by development work.

Probable ore is the ore in place that has not been completely proved by exploration work.

Possible ore is the ore in place that from the general knowledge of oreshoots, is believed to exist in partly explored shoots and in the lower depths of the present producing ones.

Formerly ore-reserve estimates were made by using the ratio of 9 cubic feet of ore in place to 1 long ton of ore. This ratio resulted in an under-estimate of the ore reserve; the ratio 8.5 is now used.

Tonnage from the major development work and from part of the stope development work is not included in the ore-reserve estimate. During 1930 this tonnage was 5.6 per cent of the total tonnage hoisted.

METHODS OF DEVELOPMENT AND MINING

Early Development and Mining

Open-pit mining was the first method used at Mt. Hope, and with the long outcrops due to the low pitch of the oreshoots, considerable tonnage was extracted by this method. Where the topography permitted, drift and crosscut adits were driven to develop the ore at lower depths, the drift adits being advanced until the cap or bottom rock was encountered. The adits drained the mining areas above the levels, which at that time was an important consideration.

The ore above the adit levels was mined by underhand stoping, 20 feet being a common stope height. The ore was blasted to the level floor and there shoveled into cars by hand. In several instances stopes open to the surface were mined downward to depths of 100 feet or more. With the exhaustion of the ore above the adits, winzes were sunk on the dip of the ore. Finally each oreshoot was developed by an independent shaft sunk on the dip of the ore. From these shafts levels were developed by drifting northward and southward in the ore.

The level interval varied from 40 to 60 feet. The level drifts were advanced until the cap or bottom rock was encountered. Mining advanced from the shaft with occasional shaft pillars, and floor pillars were seldom left. To maintain the level travel ways from the shaft to the mining faces, stulls were placed from hanging to foot wall, and on these the track was relaid. The ore was likewise here mined by underhand stoping, blasted to the next lower level, and there shoveled by hand into hand-trammed cars. All of the ore south of the Mt. Hope fault excepting that mined from the 1,000 level was mined by this underhand stoping method. The percentage of ore extracted was low, mainly due to forced abandonment of incompleted stopes on account of failure of the stope walls.

The Brown shaft, the present operating shaft, was sunk about 1890 to the bottom rock of the Taylor orebody, a depth of 500 feet. It is a 3-compartment, 64° inclined shaft. It was started in the footwall, but encountered the ore at 250 feet and continued in ore to the bottom rock. From this shaft the 300 and 400 levels were developed. From the 300 level the North and South Taylor oreshoots were mined, and from the 400 level the North and South Leonard oreshoots were mined. Considerable exploration work was conducted from these levels in search of the northward extensions of several smaller oreshoots that were mined south of the fault. It was from the 300 level that shrinkage stoping was first employed.

With the exhaustion of the ore above the 300 and 400 levels, the northern extension of the Taylor and the Leonard orebodies was developed by separate inclines driven on the downward pitch and in the bottom rock of the oreshoots. The later 1000-level development connected with these inclines.

The Taylor oreshoot lying between the 300 and the 1000 levels was developed by two intermediate levels from the incline, the level interval varying from 160 feet to 260 feet. Shrinkage stopes with intervening pillars of ore were developed from the incline and from the levels.

The shrinkage stopes were 600 feet long, the stope pillars 30 to 40 feet wide. Manways were driven in the center of the pillars from level to level. Pillar raises were not driven, entrance to the stope being through 45° raises driven from the stope to connect with the manway. Ore chutes were placed at 40-foot intervals; no stope grizzlies were used. The 600-foot stopes were too long for efficient mining.

The Leonard stopes between the 400 and the 1000 level were all developed and mined from the incline, no intermediate levels being necessary due to the lesser height of the orebody. Early development is shown in Figure 1.

1000-Level Development

The 1000 level, the one operating level from which the present mining operations and all new incline development are being conducted, was developed from the Brown shaft, which had been sunk to 100 feet below the level. At the shaft station (fig. 2), a shaft loading pocket and grizzly (fig. 3), pump station and sump, and a drill-steel sharpening shop were cut. More recently, in preparation for sinking the shaft, a hoist room (fig. 2) was cut in the footwall of the shaft and a rope raise was driven to connect with the manway compartment of the shaft through which sinking was to be conducted.

From the 1000-level shaft station, level development has been extended to develope four orebodies; three of these are being mined, and the fourth, the Carlton, due to its smaller size has been temporarily abandoned.

The 1000 level development (see fig. 1), including the inclines driven from the level, has a longitudinal extent of 7,500 feet and lies within a zone 1,000 feet wide. The level development consists primarily of 7,000 feet of drifts, driven mostly in ore, and 2,150 feet of crosscuts driven in rock. From the 1000 level a total of 1800 feet of incline has been driven upward on the pitch of the ore, and 5,450 feet on the downward pitch. Active operations are being conducted through 6350 feet of this 7,330 feet of incline development. The respective lengths of the four active inclines are 450 feet, 1,950 feet, and 2,100 feet downward from the level, and 1,370 feet upward from the level. New incline development is entirely northward on the downward pitch of the ore.

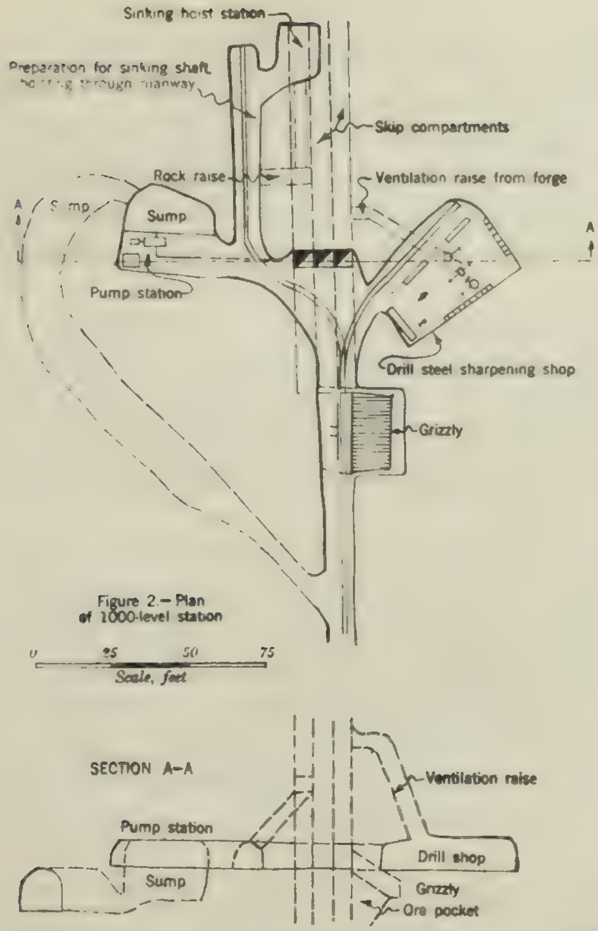

Figure 2.— Plan of 1000-level station

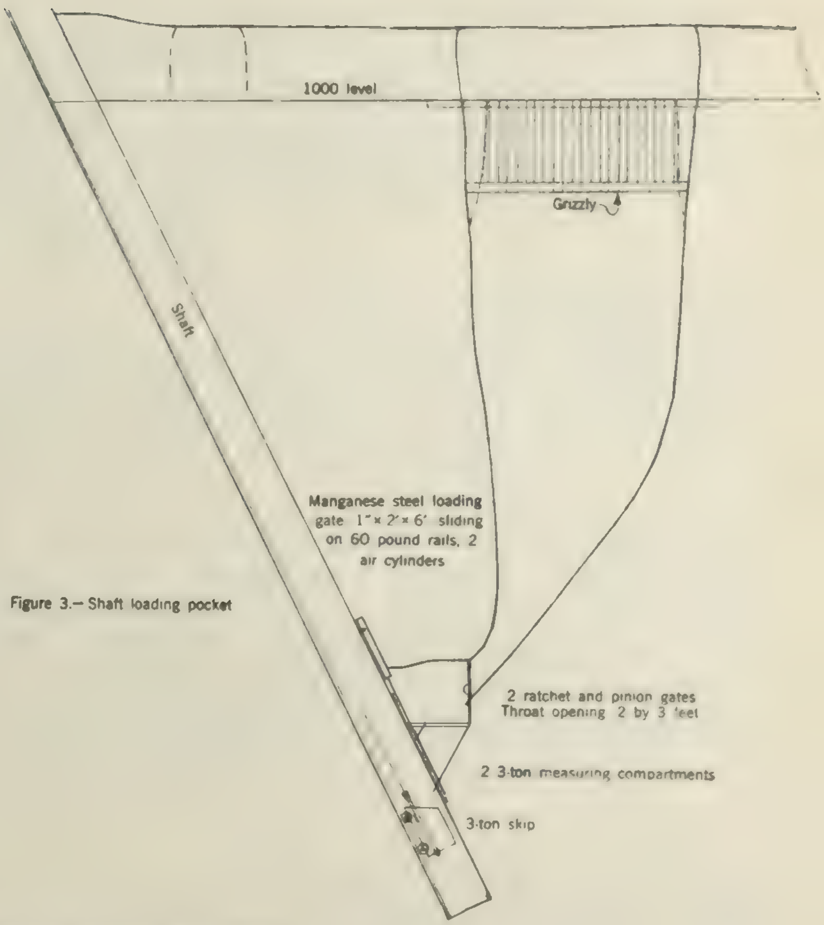

Figure 3.— Shaft loading pocket

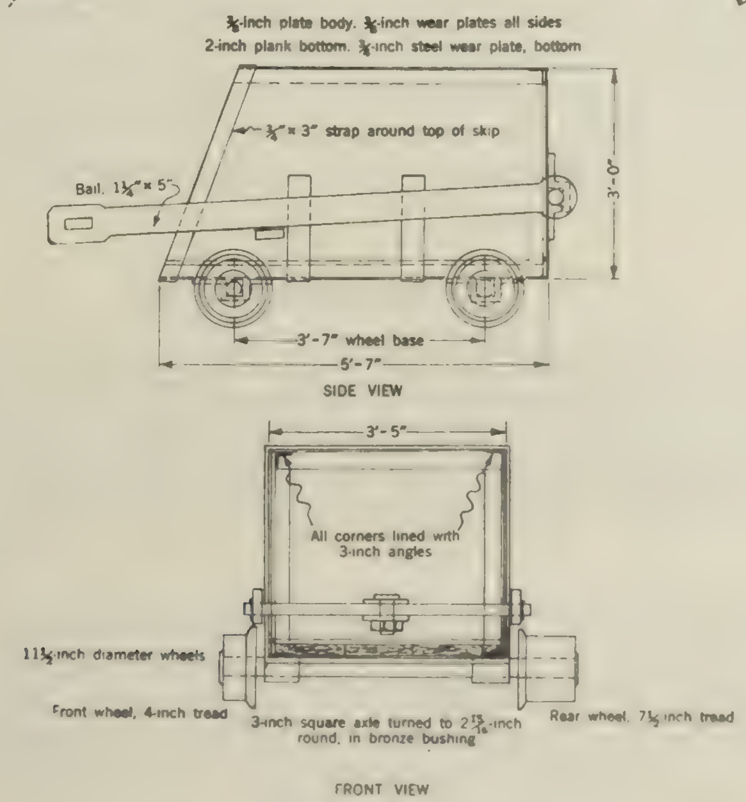

Figure 4.— Main incline shaft skip, 48-cubic foot capacity

To reach this stage of development, a crosscut was first driven eastward from the shaft station, which is located on the western edge of the ore zone and south of the Mt. Hope fault, until the strike line of the South Elizabeth oreshoot was encountered; from this point the South Elizabeth drift was driven southward. Where this drift encountered the bottom rock, an incline was driven upward and downward on the pitch of the ore.

The crosscut was then turned northward to traverse the Mt. Hope fault, and then advanced eastward paralleling the fault strike and cutting the strike lines of the North Elizabeth and of the South and North Carlton. The South Carlton drift encountered the cap rock and advanced through the ore to the bottom rock. The North Carlton drift advanced to the bottom rock of the oreshoot.

In the central part of the ore zone the main north drift was turned off from the crosscut and followed the central vein northward, passing through the North Elizabeth oreshoot and on through the Leonard to its cap rock. From this drift respective inclines were driven to develop the downward extension of the two orebodies. From the northern end of the drift a crosscut was driven westward to encounter the bottom rock of the Taylor orebody, and from that point the ore was developed by a drift driven northward to the cap rock and by an incline driven on the downward pitch of the ore.

All of the drifts followed the ore. The aim in driving the inclines is to keep them straight (variations in the angle of slope being more easily cared for than deviation to the right or left), and to keep the incline from 25 to 30 feet beneath the commercial widths of the ore above. This location of the incline permits a high percentage of extraction of the ore, and does not demand undesirably long chute raises to reach the grizzly chambers or the stope, as the case may be.

Stope Development

The general plan of stope development is the same whether started from the level or from the inclines. The length of the stopes is usually 340 feet, although there are variations from this length due to local conditions. The ore pillars between the stopes are 30 feet wide in the Taylor stopes, and from 35 to 40 feet in the stopes of the other two orebodies. Typical stope development is shown in Figure 5.

Entrance to the stopes is provided by a manway driven from the incline upward through the center of the pillar and connected to the pillar raises by short drifts. Paralleling the manway, but driven from the adjacent chute raises or grizzly chambers, the pillar or mining raises are driven, these forming the cut-off line between the stope and the pillars. If access to the stopes is possible from the level above down through the pillar raises, the manway is eliminated. In the Taylor stopes the manway and raises are driven on the footwall of the ore; in the Leonard stopes, due to the rolls in the footwall, the raises are driven near the hanging wall.

To permit freest drawing of the broken ore from the lower end of the stopes, the pillar raises should be driven in the vertical plane, but this is not possible with the system of raising used. All of the pillar raises lean downward from the vertical, causing the upper sides of the pillars to become a false footwall on which the broken ore lies. Former pillar raises driven at an angle of 55° with the horizontal caused congestion of the broken ore at the lower end of the stopes, and necessitated hand shoveling to keep the stopes open. The present raises are being driven at an angle of 62 to 65° to eliminate this trouble.

In the Taylor stopes, each stope is provided with four ore chutes at 81-foot intervals. These are belled out directly above the chutes to make room for the grizzly chambers. From the ends of the grizzly chambers, the undercutting raises are driven on a 45° angle; a pillar is left over the grizzly chambers. Figure 13 shows details of stope grizzly construction.

Entrance to the grizzly chambers is through a subincline driven in the hanging wall parallel to the main incline, at the elevation of the grizzlies, and connected to the grizzlies with short crosscuts. The subincline is generally made continuous from stope to stope, primarily to assist in the ventilation of the grizzly chambers and of the incline below.

A simpler plan of stope development has been applied to most of the Elizabeth and a number of the Leonard stopes, and is justified by the smaller tonnage developed by these stopes. The lengths of the stopes, the manways, and the pillar raises are the same; the difference in the development is that the stope grizzlies and subinclines are eliminated and the chutes are placed at 40-foot intervals, requiring eight chutes to the stope, one large chute alternating with one small chute. The smaller chutes are drawn only during the undercutting, the ore breaking, and the final clean-up of the stope.

Development Details

Shaft.- The Brown shaft, the only operating shaft, is 20 feet 8 inches by 7 feet 4 inches outside timber measurements in the upper part, but the lower part is 6 feet, 4 inches wide. It has three full-sized compartments, two skip compartments, and one manway compartment. The skip compartments are equipped with 60-pound rail laid on a 48-inch gage. The manway contains a ladderway, air and water mains, a fuel-oil supply line to the steel sharpening forge on the 1000 level (fig. 8), and the electric cables.

The shaft was sunk in ore through the lower part of the Taylor oreshoot, and for a considerable section no shaft pillars were left. Through this section of the stope the shaft timbering consisted of large oak stull sets. Later the upper 80 feet of the shaft was concreted, and a considerable part of the stull-set timbering was replaced by standard framed shaft sets. The lower and newer half of the shaft is timbered by 8 by 8 inch framed sets, placed on 6-foot centers.

Sinking costs and records are not given, as no sinking has been done in recent years.

Inclines.- The inclines are 8 feet high and from 9 to 10 feet wide. They generally advance in rock that stands well and that requires little timber for support. Framed 8 by 8 inch timber sets with back lagging are placed in those sections of the incline that pass through ore or through blocky bottom rock, and especially where the ground is weak near the stope chutes in which bulldozing is done.

A 22-hole round with a 5-hole pyramid cut is used (fig. 6). The round is drilled to break 5 feet and will average within 95 per cent of this figure over a period of several months. One hundred pounds of 40 per cent dynamite is used for the 5-foot round. Stemming made of paper cartridges filled with damp iron-ore dust is used. No wooden cartridges are employed in the cut holes, but an average of one such cartridge for each of the remaining holes is used. The detonator is placed in the third or fourth cartridge from the bottom of the hole. The cut is blasted first, if time permits, and then the balance of the round is blasted.

Two 150-pound, Leyner-type drifting machines are used in each face; they are mounted on separate vertical columns and arms of $3\frac{1}{2}$-inch diameter. The drill steel used is $1\frac{1}{4}$-inch hollow round with lugged shanks. The bit is the standard 5 and 15° double tapered cross bit. Four to five drill changes are used to the hole. The drill lengths and bit gages follow:

	Length, Feet	Length, inches	Gage, Inches
Starter	2	6	2 1/2
Starter	3	0	2 3/8
Second	4	3	2 1/4
Third	5	6	2
Fourth	6	9	1 7/8

The incline sinking crew consists of two drillers, one chuck tender, and from three to four muckers, depending upon the actual mucking time available during the eight hours. The drilling crew drills and blasts in the afternoon shift, and during the following day shift the mucking crew cleans out the face and lays the track.

Main Level Drifts.- No main level drifts or crosscuts have been driven in recent years, so costs and records will not be given. The main haulage drifts and crosscuts are 8 feet high and from 9 to 10 feet wide. Framed 10 by 10 inch drift sets with back lagging are placed throughout most of the drift lengths. Replacement of drift sets is mainly due to timber decay.

Subinclines, Pillar Drifts, and Crosscuts.- The method of driving the subinclines, the pillar drifts, and the crosscuts is the same. These openings are used in the development of the stopes. They are from 3 to 4 feet wide and 6 feet high. The crosscuts are only one to two rounds in length, the pillar drifts from 12 to 24 feet, and the subinclines are driven in sections 80 feet long. No timber is necessary for support, whether driven in ore or in rock.

The entire round is broken with one blast. Primers are made up on 8-foot fuse and No. 8 detonators. The holes are drilled wet, as are all rounds in development headings. The air hoses are 1 inch, and the water hoses are $\frac{3}{4}$ inch in diameter. Self-rotating, 100 to 110 pound wet stopers are used throughout for this development work. The headings are driven on the horizontal or on an upward slope of 5 to 18°. All holes are drilled normal or nearly normal to the face. The air feed-leg of the machine is given support by a simple set-up consisting of 2-inch plank supported by two extension legs that are made of a piece of $1\frac{1}{2}$-inch pipe with collar and set screw, and a short length of drill steel (fig. 7).

The "burnt cut" is used in all of the rounds, and requires from nine to ten holes, drilled parallel and straight in. Nineteen to 22 holes are used to the full round. The cut holes are drilled a few inches longer than the balance of the round. Of the 10 cut holes drilled, only five are loaded and blasted. The central cut hole in a 6-foot round is loaded with 10 to 11 1 1/8 by 7 inch cartridges of dynamite, those nearer to the collar being but loosely tamped. In the other four cut holes, as well as the rest of the round, from three to four wooden space cartridges are used in each hole. The primer is the third or fourth cartridge from the bottom. With this round advances of 6 feet are consistently made. Seventy pounds of 40 per cent dynamite is the average amount required for each round (fig. 6).

One miner is employed in each face. He moves his supplies and equipment from the main incline to the working face, drills and blasts his round, and cleans the face of the ground broken. When drilling in ore and little shoveling is to be done, a 6-foot round is pulled every two days. In an 18 to 24 foot pillar drift, where considerable shoveling is necessary, $2\frac{1}{2}$ to 3 days are necessary for each round. A maximum of four days is required to break and clean a round in the subinclines when the round is drilled in rock and the broken rock has to be moved with a wheelbarrow for a distance of 50 to 80 feet from the face. One 8-hour shift per day is worked in the headings.

Four to five drill lengths are used to the hole. The drill lengths and bit gages follow:

	Length		Gage
	Feet,	inches	Inches
Starter	2	6	2
Second	3	8	1 7/8
Third	4	8	1 3/4
Fourth	5	8	1 5/8
Fifth	6	8	1 1/2

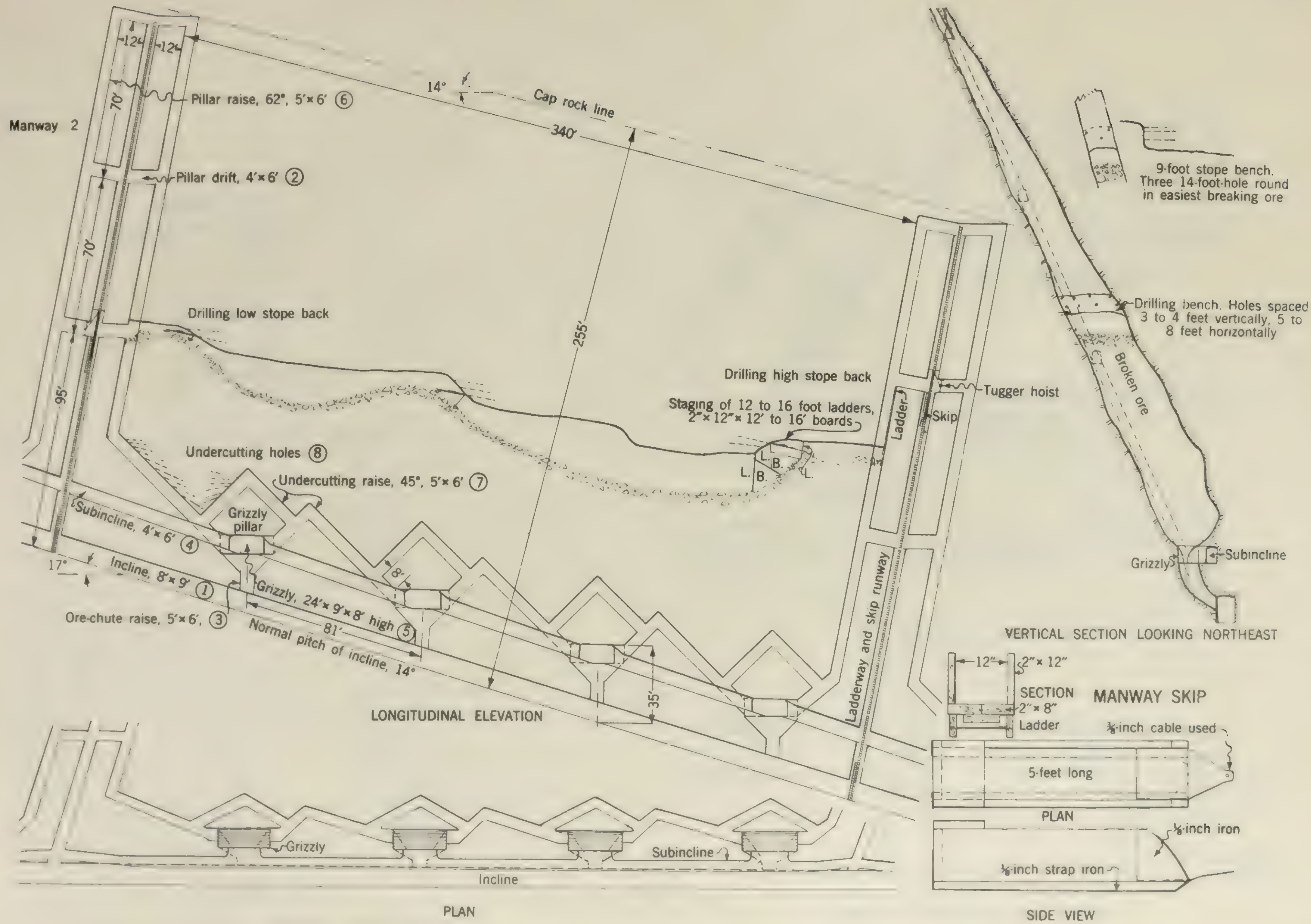

Figure 5.– Shrinkage stope development and mining. Numbers in circles indicate successive development steps

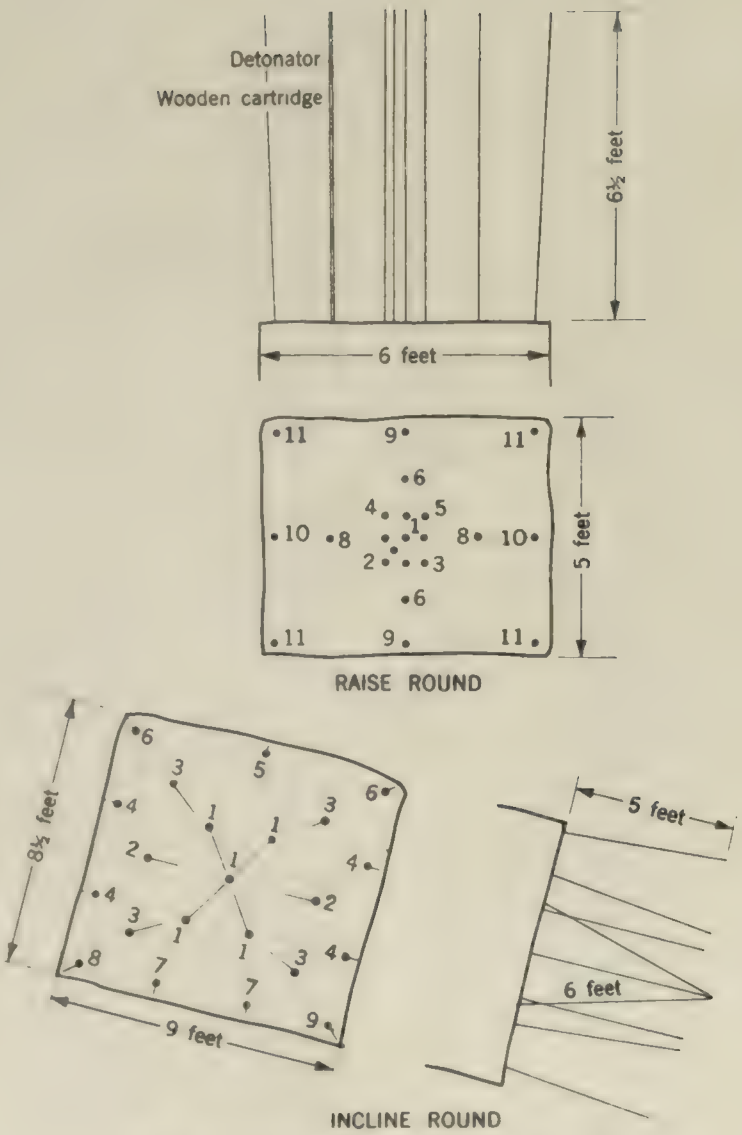

Figure 6.— Drilling rounds. Numbers indicate firing order

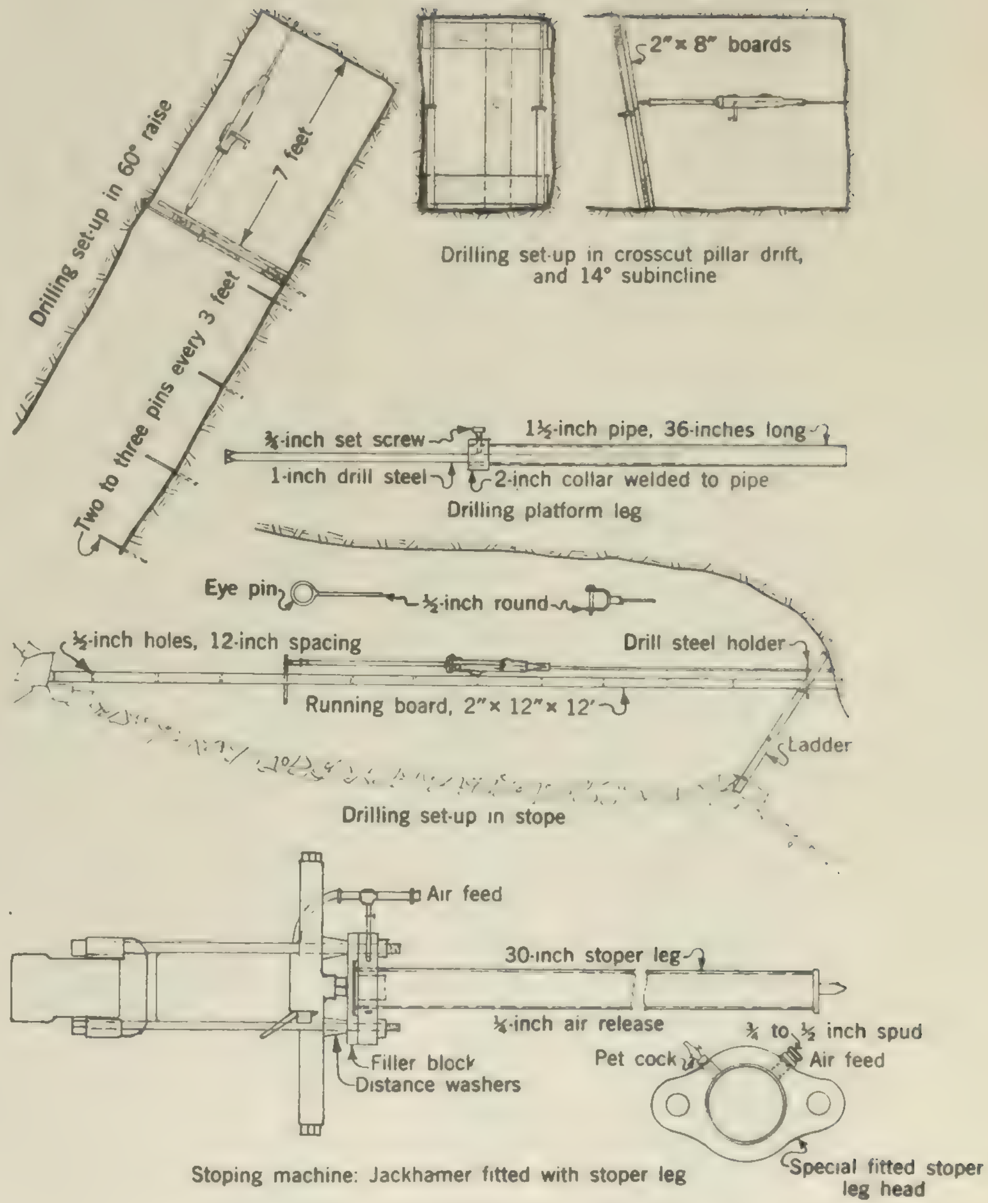

Figure 7.— Drilling equipment and set-ups

Raises.- The raises are 5 feet high and 6 feet wide; they are driven at an angle varying from 45 to 65° from the horizontal. The raises are locally classified as chute raises, undercutting raises, pillar raises, and manways. They vary in length from a few rounds as required by the stope chute raises, to over 200 feet necessary for the manways and the pillar raises. All raises are driven single compartment; no timber is necessary for support.

Access to the face of the raises is by means of pins placed along the footwall, two to three pins being placed at every 3-foot length of raise. The pins are short lengths of discarded $1\frac{1}{4}$-inch Leyner drill steel. Holes for the pins are drilled with a lightweight jackhammer. (See fig. 7.)

The longer pillar raises and manways (see fig. 5) are not driven in one life. The manway is first driven to the elevation of the first set of pillar drifts, the pillar drifts are driven from the manway, and then the work in the manway is temporarily stopped. The upper pillar raise is then driven to connect with the face of the upper pillar drift, and the raise is equipped with a ladderway and with a tugger hoist placed at the top of it. The ladderway serves as a slide for the small skip used (fig. 5). A bulkhead is then placed in the manway at the elevation of the pillar drifts, and the work in the manway is resumed. The two raises are advanced successively from one set of pillar drifts to the next set above until the cap rock of the ore is encountered. The tugger hoist is moved upward upon the completion of each lift and serves the raises with supplies and equipment from the main level. The cost of raising at a 200-foot height is not much more than the cost of the first 75 feet and is equally as safe. The cost of placing and removing the ladderways from one raise to the other is not great. The air and water pipes, which are of $1\frac{1}{2}$ and $\frac{3}{4}$-inch diameter, respectively, are carried up in the lower corner of the raises and are fastened by chains to the steel pins.

As soon as the main incline has advanced sufficiently for the next stope chute, the sinking crew drills the first round with the Leyner-type machines. The chute raise is then advanced, using wet stopers, as soon as possible to encounter the ore above. This is done to give a check on the position of the ore relative to the incline.

One man is employed in the face of each raise. He moves his equipment and supplies to the face, and drills and blasts his round.

Self-rotating 100 to 110 pound wet stopers are used. The drilling platform consists of 2-inch plank supported on extension legs, the same equipment as is used in the subinclines and crosscuts (fig. 7). For raises of greater height a set of "safety pins" are placed directly beneath the platform.

Four to five drill changes are used to the hole, one set of drills serving for more than one hole. The different drill lengths and bit gages are practically the same as used in the subinclines, with the exception that considerable drill steel is made up for drilling 7-foot rounds. The raise round (fig. 6) uses the "burnt cut" and is different from the subincline round only in that with the same number of holes the raise round breaks a larger opening.

Two days are required by one man to clean the face, set up, and drill and blast a 6-foot round. The full round is broken in one blast. But one shift, the day shift, is worked in any face. If it is necessary to blast misfires, this is done as soon as the miner comes on shift; he can still pull the round in the two days. When the raise is starting from the main incline, 7-foot rounds are drilled.

Primers are made up of 8-foot fuses and No. 8 detonators. Sixty pounds of 40 per cent gelatin dynamite is used to break the 6-foot round. The round is loaded, using wooden cartridges, in the same manner as the subinclines. No mucking is necessary in the raises.

Space Blasting.- Wooden powder is used in all development and stope rounds for space blasting. Its use has reduced the cost of powder per unit of development footage and of broken ore without resulting in a sacrifice of development footage or of tons broken per foot of hole drilled; no deleterious effects, such as excess powder gases, misfires, or partial detonation of the powder stream, have resulted.

The purpose of the wooden cartridges is to make possible the proper distribution of the dynamite along the full length of the hole to be blasted. If no dynamite is placed to within $1\frac{1}{2}$ feet of the collar of the hole, especially in the development rounds drilled in ore, the round will break inside, but will leave the collar of the holes unbroken.

The wooden cartridges are octagonal, 8 inches long, and $1\frac{1}{4}$ inches in diameter. Cartridges 12 inches long have been successfully used where light blasting was necessary and the burden on the holes was light.

Blasting Supplies.- Forty per cent gelatin dynamite is used for stoping and for all development work except in the Taylor incline, where equal weights of 40 per cent and 50 per cent dynamite are used. The dynamite cartridges are 7 inches long and 1 1/8 inches in diameter. A safety fuse and No. 8 detonators are used. All primers are made up in a central underground station on the 1000 level. No electric blasting is done in the mine. The underground powder magazine is supplied daily with the necessary amount of dynamite. The magazine is electrically heated.

SHRINKAGE STOPING

Shrinkage stoping without timbering or filling is the only stoping method used at Mt. Hope (fig. 5). The height, thickness, shape, and steep dip of the orebodies, with the fairly high strength of the ore and the strong wall rock, all favor shrinkage stoping; these favorable characteristics are necessarily exploited to the highest degree possible, with due regard to safety and the greatest economic extraction of the ore. These factors manifest themselves in the length of the stopes, the size of the stope pillars, the drilling method, the absence at times of sufficient stope blockholing, and in the number of benches being drilled in a stope at one time.

Undercutting.- The entire length of the stope is undercut from end to end, the undercutting being conducted from the undercutting raises. All undercutting is done on a 45° angle, and extended from the raises upward until the stope walls are encountered.

Undercutting generally starts in the upper end of the stope, where the incline development is downward, mainly because the lower end of the stope is not fully developed. There is an advantage in undercutting the lower end first, in that stoping can start sooner and that two crews can be employed, one at each end of the undercut section.

Wet self-rotating stopers and dry jackhammers equipped with a standard stoper leg and operated on a 2 by 12 inch plank, are used for undercutting. The wet-stoper round is drilled upward, its use being limited to widening out near the raises and cutting out above the grizzly chamber pillars. The balance of the undercutting advances from the raises working upward, the rounds being fanned outward and drilled downward. For this drilling the specially equipped jackhammer and equipment described below are used to drill 14-foot rounds.

Stoping Practice.[4]- In the Leonard and the Elizabeth stopes two crews are worked, one crew at each end of the stope and both advancing toward the middle (see fig. 5). Each crew consists of two men, each man being provided with a complete set of drilling equipment. The ground back of the advancing faces is closely watched; the area between the two crews receives but little attention except as to the proper drawing of the broken ore. Should the bench being advanced or the ground to the rear become heavy and unsafe, the crew retreats to another bench in safe ground or to the end of the stope to start a new bench; the crew advancing from the other end will advance to the bench abandoned.

In the Taylor stopes, with their stronger ore and walls, drilling can be conducted from a greater number of benches; however, five machines are all that are used at the present rate of production.

The height of the drilling benches varies from about 6 to 15 feet, and generally varies inversely to the ore width. In the weaker ore only sufficient bench is maintained to permit one days drilling, the round being blasted at the end of the shift. In the stronger Taylor ore, one bench may provide several days drilling.

Cleaning down the back of the stope and the face of the bench requires considerable time, but with the quick and flexible drilling set-up used and the relatively short time required for drilling, a good round is generally drilled and blasted daily by each crew. The two machines may drill on the same bench or on separate benches; where the ground is bad only one machine may be used.

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4 Radel, F. M., Mount Hope Drilling Method: Eng. and Min. Jour., July 24, 1930, pp. 75-79.

The present drilling method (figs. 5 and 7) has been in use for about three years; improvements are constantly being made and the miners are becoming more adept in its use and application since its inception. The method was evolved through the necessity of drilling horizontal holes from the end of a stope in which the ore had been drawn a considerable distance below the back (too far for drilling vertical holes from the top of the broken ore). Drilling with a jackhammer pushed along on top of a plank supported by staging was resorted to.

The drilling of long horizontal holes proved so effective that their use soon superseded the former method of drilling 6-foot holes vertically with stoper machines. The present method of drilling is much the safer of the two, for the miner is working under ground thoroughly cleaned and tested, and with the drilling being done ahead of him; with the vertical holes the miner worked beneath the ground being drilled, which at Mt. Hope is not safe practice.

Following are comparative stoping figures, the figures for the year 1926 being the results obtained during the last year that the old drilling method was used, and the figures for 1930 being the results obtained by the new present drilling method. The prime difference between the two methods is that the old method used vertical holes about 6 feet long drilled with wet and dry stoper machines, and the new method uses horizontal holes about 14 feet deep and drilled with jackhammers fitted with a standard stoper leg.

	1926	1930
Tons broken per machine shift	28.66	82.03
Tons per labor shift chargeable to all stoping work	- -	57.09
Man-hours per ton stoping	0.402	0.138
Average tons crude per shift underground ..	4.68	7.088
Average tons crude per shift on property ..	3.08	4.522
Tons broken per foot of hole drilled (stoping)	0.745	2.579
Pounds powder per ton broken (stoping)	0.611	0.384

Not only has the new drilling method proved more efficient, it has removed the hazard, relatively speaking, from the stoping of the ore by the shrinkage method.

Drilling Equipment.- Two-inch air lines are carried from the main level up through the manway at each end of the stope, through the pillar drifts, and down the pillar raises into the stope. Short lengths of $1\frac{1}{2}$-inch hose are used for connecting the pipe around sharp bends. Water is available through $\frac{3}{4}$-inch pipe for wet drilling but is seldom used.

A jackhammer fitted with a standard stoper leg is used for stope drilling (fig. 7). To fit the stoper leg to the jackhammer, a blind filler plate is placed between the head of the jackhammer and the head of the leg. The side rods are extended to receive the stoper leg head, distance washers being placed between the handle head and the filler plate. To supply and control the air pressure to the stoper leg, the head of the leg is drilled and tapped in two places, one for a $\frac{1}{4}$-inch petcock, and the other to receive a $\frac{1}{4}$ to $\frac{1}{2}$ inch spud. A short length of $\frac{1}{2}$-inch hose with a 3/8-inch valve is connected from the $\frac{1}{2}$-inch spud to the $1\frac{1}{2}$-inch air hose by means of a T inserted in the jackhammer air-hose connection. Air inflow to the stoper leg is through the $\frac{1}{2}$-inch hose and is controlled by means of the 3/8-inch valve. By means of the $\frac{1}{4}$-inch petcock the air pressure applied to the leg extension is controlled and is released by fully opening it. The stoper leg, when the extension is fixed in place, and the air pressure is applied and kept under control by means of the petcock, becomes the drill feeding device.

Stoping drill steel is made up from 1-inch hollow hexagonal steel. Formerly the steel was not shanked, but it is now being gradually changed over to have a collar shank. The drill lengths and bit gages follow:

Steel	Length, feet	Gage, Inches
Starter	5	2
Second	7	1 7/8
Third	11	1 3/4
Fourth	14	1 5/8

Some 16-foot drill lengths are used.

Stope Drilling and Blasting.- The drilling machine is placed upon a 3 by 12 inch plank, 12 to 16 feet long. The plank is supported at one end by simple staging made of one or more short ladder lengths and of 2-inch plank. The foot of the drill extension leg is held in place by means of an eye pin inserted in $\frac{1}{2}$-inch holes bored along the center of the plank. As the drill hole advances and the leg extension is run out, the pin is moved forward accordingly. The drill is held in place for starting the hole by means of a simple accessory placed near the end of the plank. The jackhammer used must have a strong air-blowing device to clear the drill cuttings from the bottom of the hole. Difficulty is encountered in keeping the hole clean when drilling with water.

With the above drilling equipment, holes can be drilled where the back room is not more than 2 feet, as is the case along the footwall at times; or by means of the ladder-plank staging, 15-foot backs can be easily and safely drilled.

The amount of burden on a hole varies from 3 feet to about 6 feet vertically. Ore widths of from 4 to 10 feet are broken with one hole placed midway between the walls and cleaned with two holes, one along each wall, 4 to 5 feet above the bottom hole. The average stope round is drilled 14 feet deep and nearly horizontal. Considerable overbreak ahead is obtained.

Forty per cent dynamite in cartridges 1 1/8 by 7 inches long is used for stoping. The primers are made up of 8 to 10 foot fuse and No. 8 detonators. Wooden spacing cartridges are used in all of the holes. The 14-foot hole is loaded by first tamping 12 to 15 cartridges of dynamite into the bottom of the hole and then alternating with one cartridge of dynamite and one of wood back to near the collar of the hole. Without using the wooden cartridges and using the same amount of dynamite tamped in place, there will remain a considerable section near the collar without any explosive, and when blasted the hole will cut off inside. To string the dynamite along the hole to the collar without tamping it is not possible in holes looking upward. To place more burden on the hole and load it to the collar with explosive, results in breaking excessively large blocks of ore. Loading the bottom of the hole with 40 per cent dynamine and the balance of the hole with explosive of lower strength is not considered an alternative of using the wooden cartridges.

Blasting supplies and stope equipment are handled through the manways by means of a small skip and an air tugger hoist. The miners handle their supplies and equipment, lay the necessary pipe lines in the stope, clean down the back, and drill and blast their own rounds.

A limited amount of mucking has at times been necessary to keep the lower end of the stope open; this mucking is being eliminated by driving the pillar raises at a steeper angle.

No timber is required for support in the stope. The back of the stope is generally arched to give it additional strength.

One man is employed on a stope grizzly to blockhole the bowlders and to assist the running of the ore through the grizzly. All large blocks of ore that can be reached are blockholed in the stope. Light-weight jackhammers are used for blockholing.

PERCENTAGE OF EXTRACTION

It is the intent to recover all the ore except (1) that left as pillars between the stopes, (2) as toe pillars where the inclines leave the level, and (3) a small percentage contained in the stubs formed by the undercutting raises and in the stope grizzly chamber pillars. It is very probable that a considerable part of these latter pillars will ultimately be recovered.

The present mining method permits about 90 per cent of recovery. A greater tonnage has been extracted from the stopes than was estimated, the difference probably being due partly to underestimates and partly to dilution.

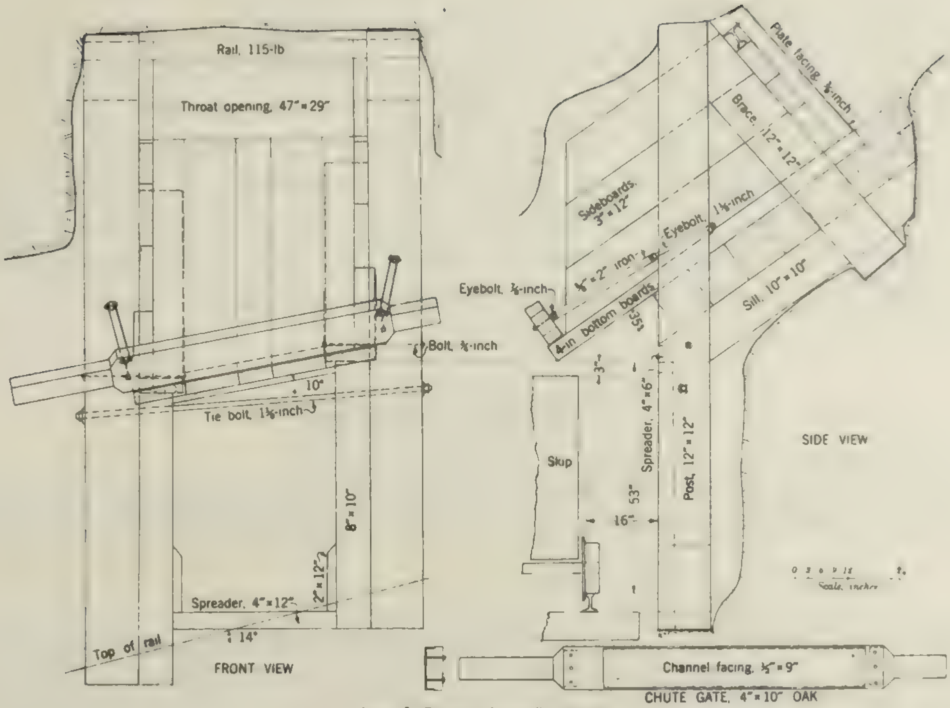

Figure 8.—Timbering for small ore chute

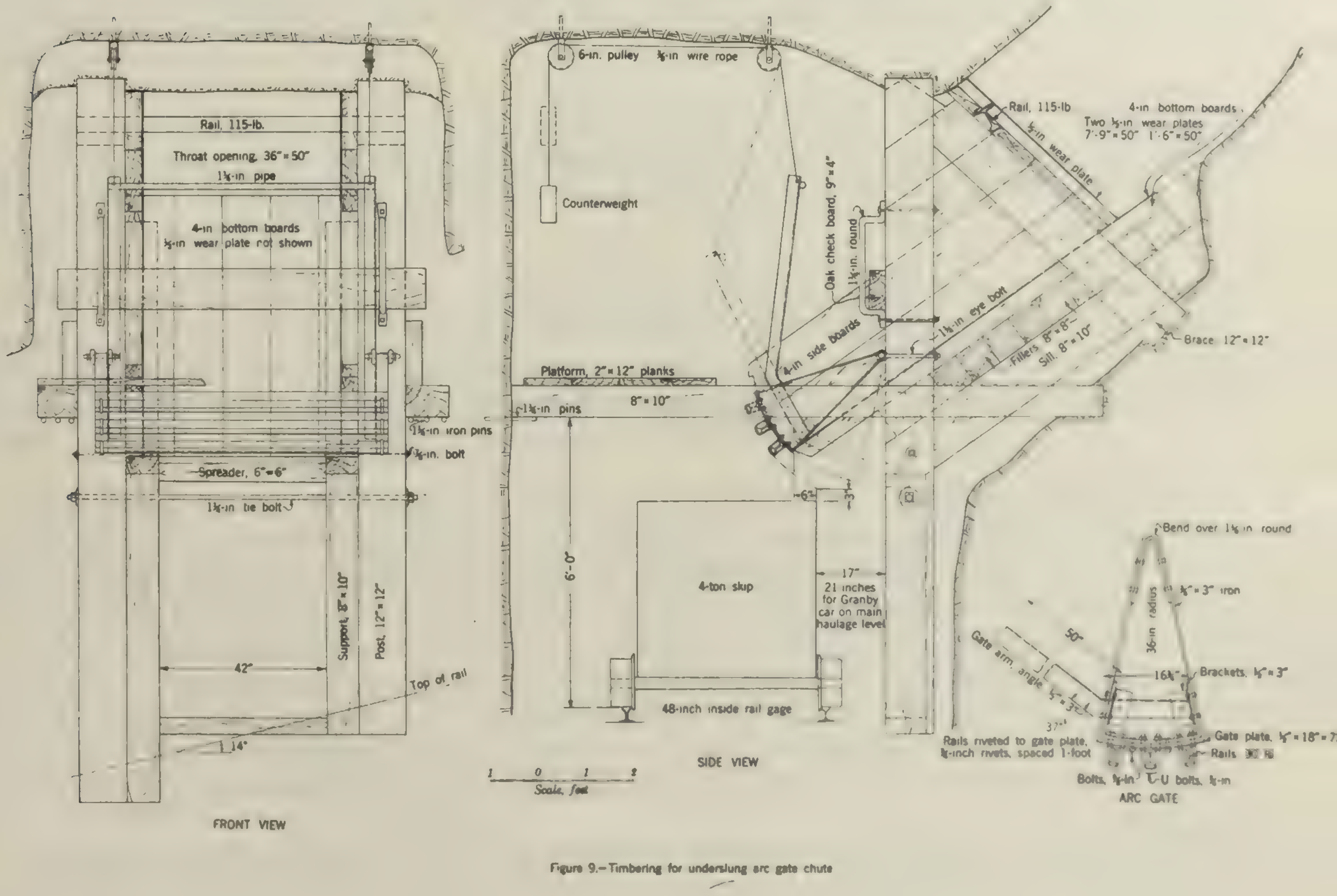

Figure 9.—Timbering for underslung arc gate chute

The dilution of the ore with waste is probably less than 10 per cent, 5 per cent of which may be attributed to intentional breaking of "vein matter" that lies on the stope walls, which if left forms a weak stope back, and the other 5 per cent to sloughing of the hanging wall and of the back of the stope after mining is completed. Waste from only one incline development can be diverted and dumped into old stopes, and the balance is hoisted with the ore; however, since the rock from the mill has a ready market for road metal and other uses, this dilution from development waste is not as undesirable as it would otherwise be.

UNDERGROUND HAULAGE

Chutes and Chute Gates.- Two types of chutes having different types of gates are used in the mine; they are referred to as the small and the large chute.

The small chute has a smaller throat opening, it does not have a loading platform, and is less costly to build than the large chute. The gate is of framed oak timber faced with channel iron. It swings upward on gate arms that are bolted to the chute posts. Two men are required to operate the gate effectively.

The small chutes (fig. 8) are used in the manways, and are removed upon the completion of the raise, and in general where the tonnage does not warrant a large chute, or where by alternating a large chute with a small chute the tonnage is adequately cared for.

The large chute (fig. 9) used on inclines and levels and for incline ore pockets is provided with a platform from which the chute-puller works. The gate is an underslung 18-inch arc gate, made of steel plate and reinforced with 32-pound rail. Both types of chutes are strongly built to withstand blockholing, and with minor repairs last about three years.

The large chute for loading Granby cars from the incline pockets at the haulage level is used exclusively in the Taylor stopes, excepting where the toe pillar is being recovered in two of the Leonard stopes; in the balance of the stopes a large chute is alternated with a small chute. The larger chute is by far the better type; it permits easier, faster, and safer drawing of the ore, and the large blocks of ore are easily reached for blockholing or bulldozing. Stope grizzlies are not used above all of the chutes (fig. 13). One man can operate the arc gate, but two men, one on the platform and the other on the level, are generally used to expedite the loading of the car or skip.

Locomotive Haulage.- The haulage ways have an average of $\frac{1}{4}$ per cent favorable gradient. The maximum haulage distance is 4,100 feet, and the average haul is 2,650 feet. Trolley-type electric locomotives are used. The line voltage is 250 volts, both rails are bonded, the rail weight is 32 pounds, and the track gage is 26 inches. The minimum radius curve is 40 feet. Stub point switches are used throughout. A ground-level type of throw switch is used; the arm parallels the track and requires but little space or clearance between the track and the side of the haulageway.

One 6-ton, one $6\frac{1}{2}$-ton, and two 4-ton locomotives are employed. One 4-ton locomotive services the level and the two larger ones are used for ore haulage, with the other 4-ton available if necessary. They are all driven with gear reduction by two motors.

Mine Car.- Granby-type mine cars[5] are used for ore transportation on the level (fig. 10). The weight of the cars is 7,250 pounds; they are fitted with Timken roller bearings. The average car load is 4.2 tons. The train crew generally consists of two men, but if necessary one man can operate the train and dump the load at the shaft ore pocket. With normal training two men are required to keep the ore pocket grizzly clean.

After three years of service, few repairs and only minor changes in design have been necessary. Trouble was experienced with the shaft of the dumping wheel, which bent just inside of the wheel. This was remedied by cutting a $\frac{1}{2}$-inch counterbore $\frac{1}{2}$ inch deep from the inside of the wheel, and then turning the shaft down with a collar to fit in the counterbore. No trouble has been experienced with the dump wheel wearing out of round. Figure 11 shows the car wheel assembly. In several of the cars the main body channel section on the hinged side of the car has failed, the flange of the channel breaking from the main section. This was repaired by placing a $\frac{1}{4}$-inch iron strap under and along the length of the channel section.

The dumping frame (fig. 12) for the Granby cars is faced with iron plate to protect it from the grizzly blasting. The frame can readily be swung by means of the hinges down and back from the track, permitting protruding loads to pass over the track in front of it. A 12 by 12 inch timber is placed along in front of the dumping car to check any car that might turn over while being dumped; to date this precaution has not been necessary.

Incline Hoisting.- The slopes of the inclines vary from 13 to 25°, 15° being an average. The incline haul varies from 600 to 2,000 feet. In one incline the ore is lowered down the incline, and in the other three it is hoisted from below. The inclines are equipped with 56-pound rail laid on a 48-inch gage, with electric bell signal system, and with electric lighting.

The incline in which the ore is lowered is equipped with a 75-hp., d.c. variable-speed motor, placed at the top of the incline. The skip has a swinging self-dumping door, the dumping of the skip being from the lower end. It is dumped into an ore pocket cut out directly beneath the incline, from which the ore is pulled into Granby cars on the 1,000-foot level.

The other three inclines serve operations below the level; each incline is extended above the level to allow room for an ore pocket between it and the main haulage level (fig. 14). In two instances the hoist stations are cut at the elevation of the main level, with a rope raise connecting to the top of the incline; in the other incline, the hoist station is cut at the elevation of the top of the incline. The former method is the preferable.

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5 Hubbel, A. E., Mine Cars: Eng. and Min. Jour., Sept. 8, 1930, pp. 225-228.

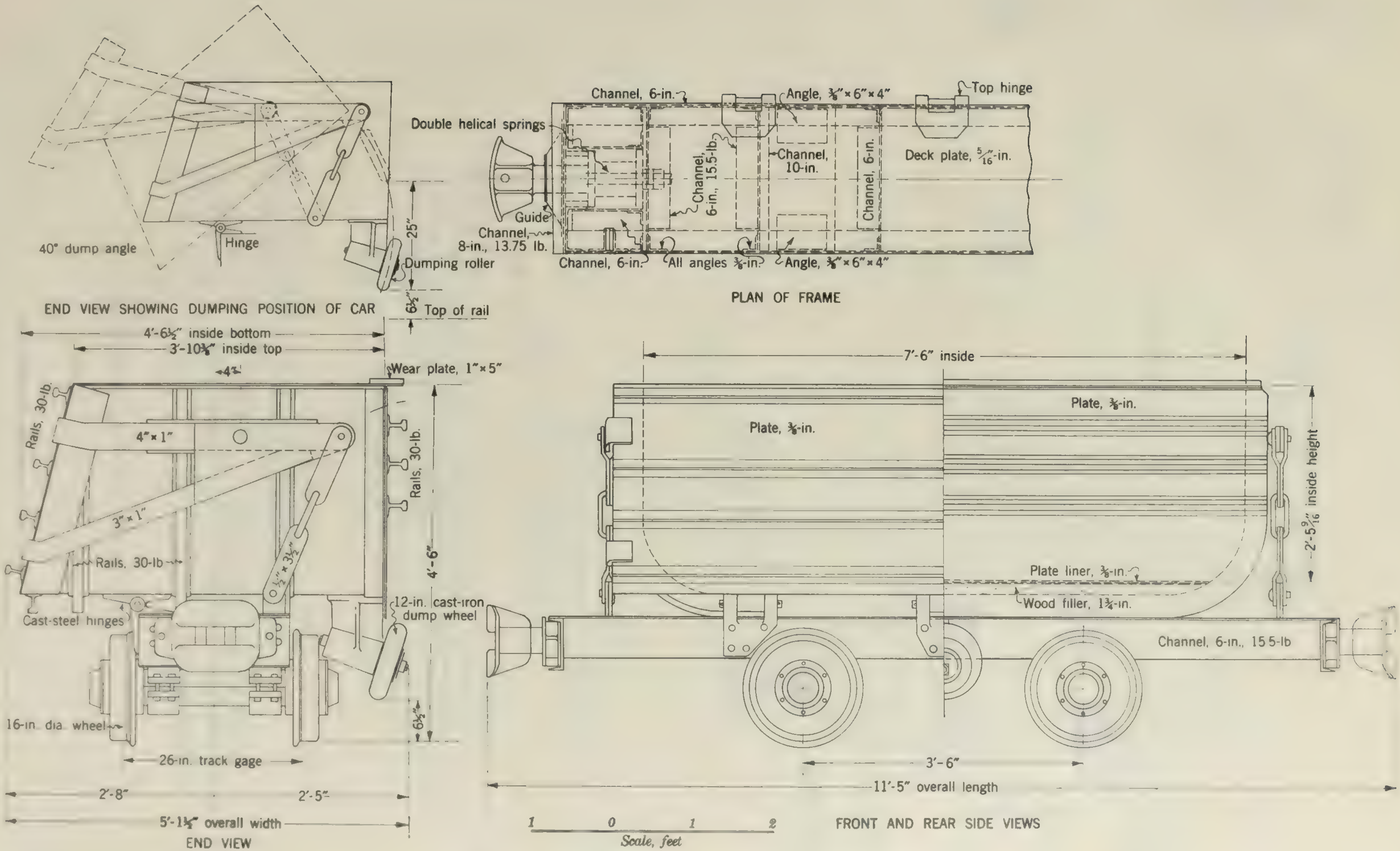

Figure 10.—Mt. Hope Granby type of mine car, 81 cubic feet

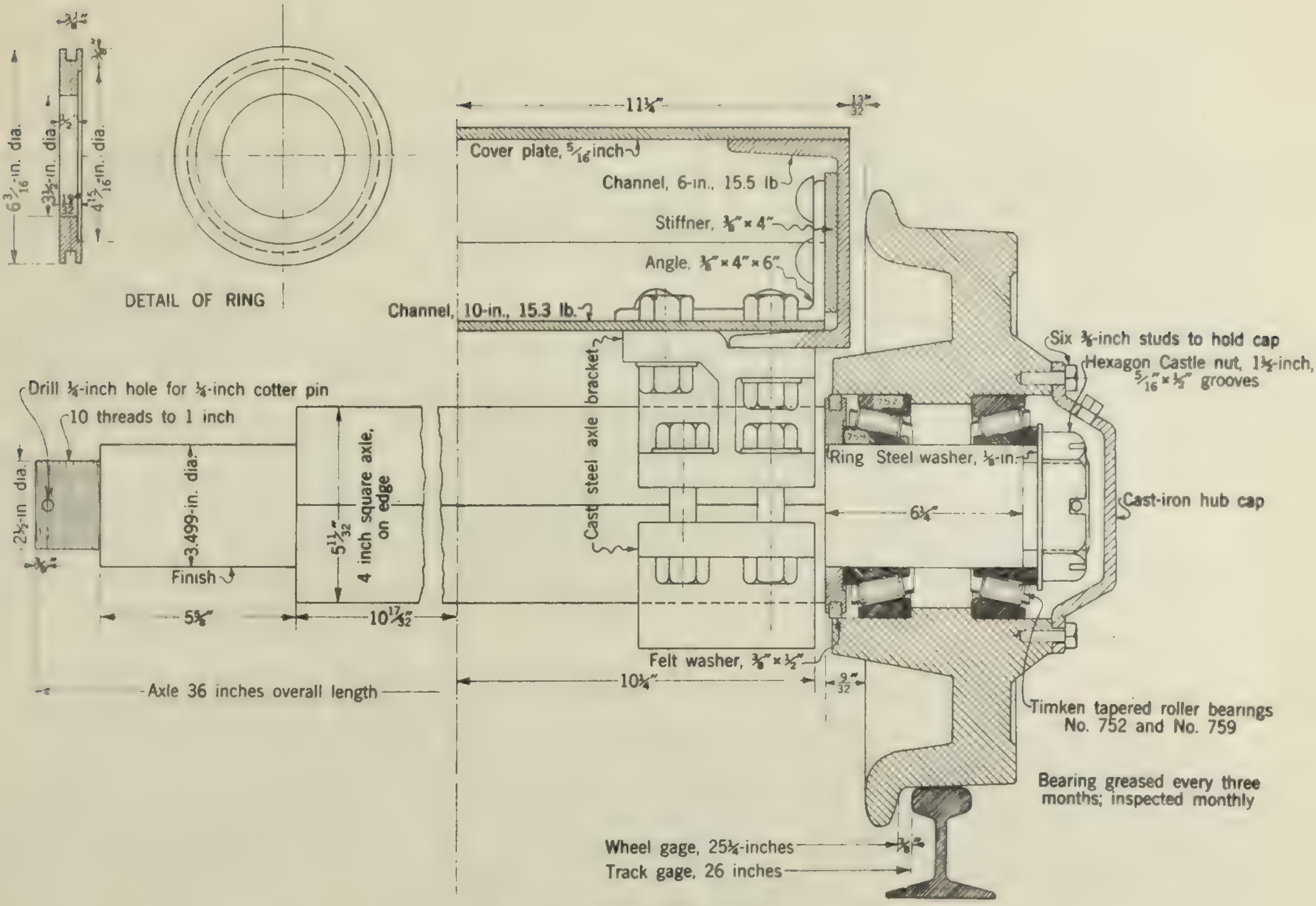

Figure 11.—Granby car wheel assembly

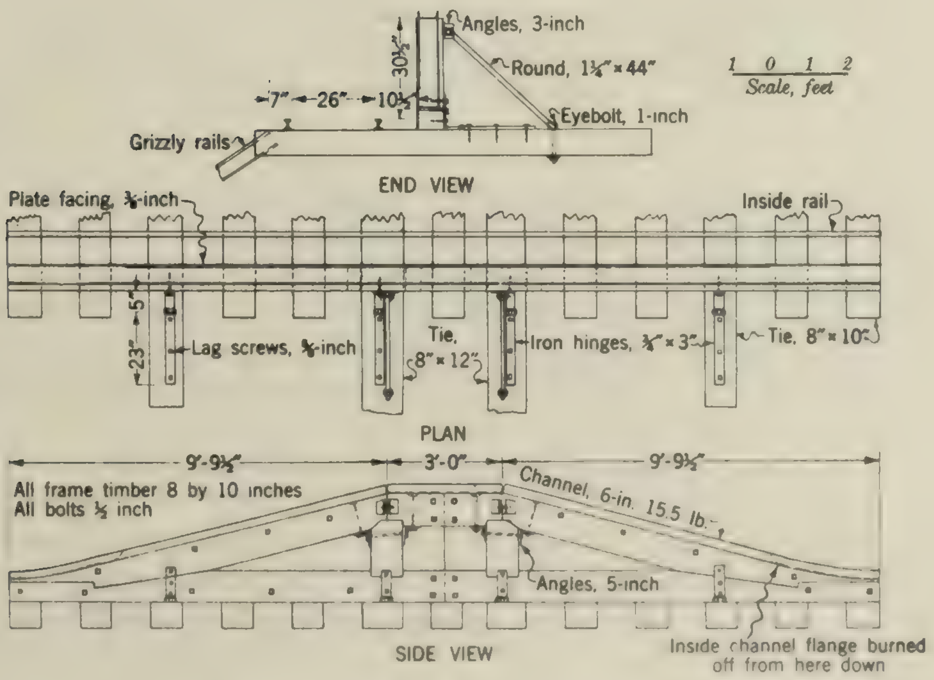

Figure 12.—Dumping frame for Mt. Hope Granby-type car

The incline hoists are single drum, d.c. motor driven, two having magnetic breaking, and are driven by 75, 100, and 200 hp. motors, respectively. Rope speeds are from about 500 to about 600 feet per minute.

In design, the incline skips are similar to the main shaft skip (fig. 4). They each have a 2-inch plank bottom with a $\frac{3}{4}$-inch bottom wear plate; the sides and the end are lined with $\frac{1}{2}$-inch plate, and have additional rail reinforcement on the upper part of the side receiving the impact of the ore as it is drawn from the chutes. The skips have swinging doors which open as the rear end of the skip is elevated when drawn on the pocket dumping frame. The bail of the skip holds the door closed when in hoisting position. The skips vary in capacity from 64 cubic feet, with a wheel base of 4 feet 8 inches and weight of 5,000 pounds, to 100 cubic feet with a wheel base of 5 feet 6 inches and weight of 8,100 pounds.

One and one-eighth inch cable is used for the incline hoisting. Rollers made of 6-inch pipe are placed in the center of the track, and where the incline has bends in it, nearly horizontal 8-inch diameter sheave wheels and vertical or sloping rollers are placed around the bend to guide the rope. To pass the rope over a rail, a spring leaf is placed along the inside of the track, the top of which is made level with the top of the rail, the spring being readily depressed when the skip wheels pass over it.

<u>Main Shaft Hoisting</u>.- Main shaft hoisting is done in balance, two 48-cubic foot skips being used (fig. 4). The hoist is gear-driven by two 550-volt, d.c., 125-hp. motors; it is equipped with air-operated post breaks and with a Lilly control. The hoisting depth is 1,200 feet, and the rope speed about 700 feet per minute; 1 1/8-inch 6 by 19 plow-steel cable is used. The life of the wire ropes is about nine months.

Two men are used to load the skips. The loading pocket is provided with measuring compartments, each compartment holding one skip load. Rack-and-pinion gates are used between the ore pocket and the measuring compartment; the skip loading gate is made up of a manganese steel plate attached to a piston rod and actuated by an air cylinder piston. The gate works on a 64° angle, sliding on two steel rails, one placed at each side.

WAGE, CONTRACT, AND BONUS SYSTEMS

Practically all men employed underground work under a contract or a bonus system. The company guarantees day wages. The present wage rate, effective since July, 1931, and which is a 10 per cent reduction from the previous wage rate, is as follows:

Muckers, drill-sharpener helpers	$4.16
Pumpmen	4.24
Miners, timbermen, chute pullers, grizzlymen, skip loaders, powdermen, motormen, pipe-fitter's helpers, incline hoistmen	4.68
Lead pipe fitter, drill-machine repairman	4.88
Lead drill sharpener, main-shaft hoistmen	5.20

Contracts

All development work is contracted on a rate per foot basis, the contractor paying for the blasting supplies. The rates for headings advancing in rock are slightly higher than when in ore. Sliding scales for raises are based on the raise height; for pillar drifts, subinclines, and for crosscuts the rate depends on the distance the rock has to be moved and the height of the working face above the level. The incline sinking rates vary slightly with increased depth and with change of ground. The contracts are closed on the sixteenth and the first of the month. The incline crew consists of 7 to 8 men, and all participate equally in the contract earnings; all of the other development contracts are one-man contracts. The present contract rates follow:

	Rate, per foot
Raises	$ 3.00 to $ 4.75
Pillar drifts	4.25 to 5.00
Crosscuts	4.00 to 4.25
Subinclines	4.00 to 6.25
Sinking main inclines ...	11.75 to 12.50

Bonus Systems

Stoping, underground ore transportation, and drill-steel sharpening are conducted under bonus systems, in all cases the bonus being figured to a per diem rate. The stoping bonus is based upon the tons of ore broken as determined by monthly stope surveys.

The underground ore transportation bonus is based on the monthly tonnage hoisted. All men employed in moving the ore from the stope grizzlies to the main shaft ore pocket, excepting the hoistmen, participate in the bonus. The drill-steel sharpening bonus is based upon the number of steels sharpened and shanked.

About 90 per cent of the underground crew are employed under the miners' rate, which at present (1931) is $4.68 per day. Approximately 17 per cent receive days wages, 17 per cent participate in contract earnings, and 66 per cent receive bonus.

During the months of August, September, and October, 1931, the contract miners earned from $5.75 to $7.57 per shift, averaging $6.65 or 42 per cent above days wages. The stoping bonus varied from 40 to 60 cents per shift; the drill sharpening bonus averaged about 80 cents per shift. The ore transportation bonus varied from 20 to 30 cents per shift.

No labor trouble has been experienced for a number of years. The labor turnover is practically nil, as the average period of employee service is not less than 10 years.

PUMPING AND VENTILATION

Pumping

During 1930 the mine made an average of 420 gallons of water per minute. The mine water varies with the seasonal rainfall. The water can usually be handled by pumping two shifts, six days a week; at times it is necessary to pump three shifts seven days a week. The water is pumped from the mine through the main shaft in two 500-foot lifts. Eighty per cent of the mine water is caught on the upper levels and drained to the 500-level pump station. All of the water made on the 1000 level and in the inclines driven from that level is drained to the 1000-level pump station.

The 500-level pump station is equipped with a 100-hp., a.c. motor, gear-driven, 610 g.p.m., vertical quintuplex plunger pump, and with a second unit, consisting of a 100-hp., a.c. motor-driven, 1,750 r.p.m., 480 g.p.m., four-stage centrifugal pump.

The 1000-level station is equipped with a 50-hp., a.c. motor, gear-driven, 300 g.p.m., vertical triplex plunger pump.

The amount of water made in the inclines is small, and is handled by 5 and 7½ hp. d.c. motor-driven, enclosed self-oiling duplex piston, 22 g.p.m. pumps. As the faces of the inclines advance, the pumps are moved downward. Small air-driven simplex piston pumps are used for pumping from the incline sumps to the small electrically driven pumps.

Ventilation

The mine as a whole is well ventilated by natural draft. The air enters the mine through old surface workings, passes through the mine, and discharges up the main shaft to the surface. One 5-hp. d.c. motor driven fan, blowing through a 12-inch canvas tubing is used to aid the ventilation of the south drift. Smoke and gases from blockholing on the stope grizzlies and in the incline stope chutes present the main ventilation problem; high-pressure air is used to disperse the smoke and gases. The sequence of stope development is to aid in the ventilation of the faces.

FIRE HAZARDS AND SAFETY METHODS

Fire Hazards

The fire hazards at the Mt. Hope mine are few. The most likely source of fire underground would be from electric cables, but this hazard is slight, due to the small amount of timber used. The upper part of the shaft is concreted and the rest is timbered; the greater part of the shaft is wet. The levels are either damp or wet; the stopes are dry, but no timber is used in them. The mine is kept clean, all refuse being hoisted to the surface. Pyrene fire extinguishers are placed in the hoist and pump stations and in the powder magazine. A secondary exit from underground to the surface is maintained.

On the surface fire hydrants connected to a 40,000-gallon supply tank are located at various points. Hose-reel carts are housed at three different places. The fire department of the town of Rockaway is available at the property within five minutes notice. All buildings are equipped with chemical fire extinguishers. The underground, surface, and district telephone systems are interconnected.

Safety Methods

Special attention is given to making the underground operations safe. By means of descriptive posters, but mainly by personal advice from the mine foreman and the shift bosses, the idea of "safety first" is kept uppermost in the minds of the employees. Safety recommendations from the men are encouraged, are heeded, and investigated. All underground employees are required to wear hard hats or caps, and hard-capped shoes. First-aid kits are maintained throughout the mine.

ADMINISTRATIVE ORGANIZATION

The administrative organization at the mine is headed by the mine superintendent, who has charge of operations. Engineering, assaying, and accounting departments are maintained; the head accounting department is at the company's offices at Phillipsburg.

The mine foreman has charge of all underground operations; he has two day shift bosses, and one night trammer boss. A surface construction foreman, a master mechanic, and a chief electrician have charge of their respective departments. The supply house is in charge of a clerk.

The total number of men employed, including the mine superintendent and others serving in supervisory capacities, were distributed as follows in November, 1931:

	Number of men
Office staff	5
Mill and surface ore transportation.	14
Surface shops and power plant	14
General surface	4
Mine (20 men on night tramming shift)	76
Total	113

ELECTRIC POWER

The Mt. Hope mine purchases its electric power from the New Jersey Power & Light Co. The power is purchased under contract and billed on two charges: energy charge and demand charge. The energy and demand charges follow:

Demand Charge

		Cost per kilowatt
First 20 kilowatts	....	$2.50
Next 30 do	....	2.00
Next 50 do	....	1.75
Next 900 do	....	1.50
Excess of 1000 do	....	1.25

Energy Charges

		Cost per kilowatt-hour, cents
First 2,500 kilowatt-hours	..	2.75
Next 3,500 do	..	2.30
Next 4,000 do	..	1.90
Next 10,000 do	..	1.60
Next 50,000 do	..	1.30
Next 80,000 do	..	1.20
Excess of 150,000 do	..	0.90

The power company supplied and installed the 750-kilowatt motor generator set and the attendent equipment, and the 33,000 to 3200 volt transformers. All equipment excepting the transformers revert to the mining company at the expiration of the 5-year contract period.

The demand in kilowatts is the highest average number of kilowatts taken during any 15 consecutive minutes of the billing month.

The power factor clause provides that when the power factor is less than 80 per cent or more than 90 per cent, the billing demand shall be increased or decreased in respect to the ratio that 85 per cent bears to the average of the monthly power factor. The power factor at the mine is about 99.8.

The daily power consumption is fairly constant. For the month of October, 1931, the power consumption, distribution, and cost follow:

Power consumption, distribution, and costs for October, 1931

Power consumption ... 217,000 kilowatt-hours.

Converted to d.c. ... 84,500 kilowatt-hours (74 per cent conversion efficiency; 39 per cent of total consumption)

Mill and Surface Consumption:

	Per cent of total
House and plant lights and shops	4.02
Primary ore crusher	2.08
Secondary crushing and concentration	11.97
Mine Consumption:	
Main shaft hoisting	12.75
Pumping	13.19
Incline hoisting and level haulage	20.97
Compressed air	35.02
Total power consumption	100.00

Costs:

Energy charge 217,000 kilowatt-hours		$2607.25
Demand charge 760 kilowatts		1187.50
Total power cost		$3794.75
Cost per kilowatt-hour	cents	1.7407
Cost per ton of crude hoisted	do	31.629
D.c. power per ton of crude ..	kilowatt-hours	7.06
For air compression per ton of crude	do	6.35

Table 1.- SUMMARY OF COSTS

Name of mine: Mt. Hope Period covered: Year 1930

Ore hoisted during period: 176,474 long tons.

Mining method: Shrinkage stoping.

Underground costs per ton of ore hoisted

	1 Labor	2 Super- vision	3 Compressed-air drills and steel	4 Power cost[1]	5 Explo- sives	6 Timber	7 Other supplies	8 Total
Incline development[5]	$0.057	$0.001	$0.027	-	$0.013	$0.002	$0.004	$0.104
Stope development[6]	.106	.004	.110	-	.042	.012	.024	.298
Mining	.074	.029	.090	-	.038	-	.002	.233
Blockholing[2]	.043	.003	-	-	.048	-	.007	.101
Transportation underground	.316	.012	-	$0.093	-	.004	.034	.459
General under- ground expense	.015	.012	-	[3] .072	-	.009	.003	.111
Total cost	0.611	0.061	0.227	0.165	0.141	0.027	0.074	1.306
Surface expense directly applicable to underground operation[4]								$0.113

1 Some supplies included.
2 Includes grizzly repair.
3 Pumping charge.
4 Includes respective percentages of workmen's compensation, general administration, surface shops, etc.; all job work is charged directly to respective mining accounts.
5 Based upon the unit costs per foot and applied to the linear footage necessary to develop the tonnage hoisted.
6 Based upon the unit costs of one representative stope.

Table 2.- SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Name of mine: Mt. Hope. Period covered: Year 1930.

Tons of ore mined and hoisted: 176,474 long tons.

Mining method: Shrinkage stoping.

	Development	Mining (Stoping)	Total
A. Labor (man-hours per ton):			
Breaking (drilling and blasting) ..	0.221[1/]	0.115	0.336
Timbering	.045	- -	.045
Shovelling	.062	- -	.062
Haulage and hoisting	.013	.405	.418
Supervision	- -	- -	.053
General	- -	- -	.215
Total underground labor	- -	- -	1.129
Average tons per man per shift ...	- -	- -	7.086
Average tons per stope machine shift	- -	82.53	- -
Labor percentage of total cost ...	12.48	8.96	21.44
Average tons per man-shift on surface properly chargeable to underground operation, including mine staff	- -	- -	105
B. Power and Supplies:			
Explosives (lbs. per ton):			
40 per cent	0.382	0.302	0.684
50 do	.020	.012	.032
Blockholing and (40 do	- -	.426	.426
bulldozing (50 do	- -	.005	.005
Total explosives	.402	.745	1.147
Total power (kilowatt-hours per ton):			13.327
1. Incline hoisting and tramming	- -	- -	2.126
2. Main shaft hoisting	- -	- -	2.294
3. Air compression	- -	- -	6.136
4. Pumping	- -	- -	2.328
5. Lighting	- -	- -	0.443
Other supplies in percentage of total supplies and power[2/]	11.38	3.66[3/]	15.04
Supplies and power, percentage of total cost[2/]	9.26	10.34[3/]	19.60
C. Percentage of Total Cost:	30.78	25.57	56.35

1/ Includes stope undercutting.
2/ Includes compressed-air power but not drilling supplies.
3/ Includes blockholing and bulldozing.

Table 3.- DETAIL OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

DEVELOPMENT

	Incline sinking	Pillar drifting	Cross-cutting	Raising	Total, all development
Linear feet[1]	808.7	200.5	42.5	2665	3716.7
Size of excavation	9x10	$3\frac{1}{2}$x6	$3\frac{1}{2}$x6	5x6	--
Timbered or not	No	No	No	No	No
Physical properties of rock	Hard	Hard	Hard	Hard	Hard
A. Labor Cost per Foot:					
Breaking (drilling and blasting)	$5.235	$4.955	$2.828	$3.289	$3.798
Timbering	.369	.009	.000	.095	.149
Shoveling	4.058	.274	.000	.265	1.088
Haulage and hoisting	2.102	.388	.056	.263	.667
Supervision	.221	.159	.126	.075	.112
Total labor	11.985	5.785	3.010	3.987	5.814
Feet per 8-hour shift[2]	2.4	3.	3.	3.	- -
B. Power and Supplies Cost per Foot:					
Explosives	2.169	1.946	0.988	1.421	1.684
Timber	.193	.126	.000	.063	.094
Total power:	1.724	1.178	1.292	1.388	1.411
1. Air compression	1.376	1.156	1.263	1.339	1.337
2. Hoisting	.315	.017	.021	.028	.020
3. Haulage	.033	.005	.008	.021	.054
Other supplies	3.650	.605	.488	1.539	1.898
C. Labor, per cent of total cost ..	60.77	60.01	52.09	47.48	48.90
Power and Supplies, per cent of total cost	39.23	39.99	47.91	52.52	51.10

1 Not including cubic footage converted to linear footage.
2 Incline sinking: 4.8 feet, two 8-hour shifts, total 6 to 7 man shifts.
Balance of development work: 6 feet, two 8-hour shifts, 2 man shifts.

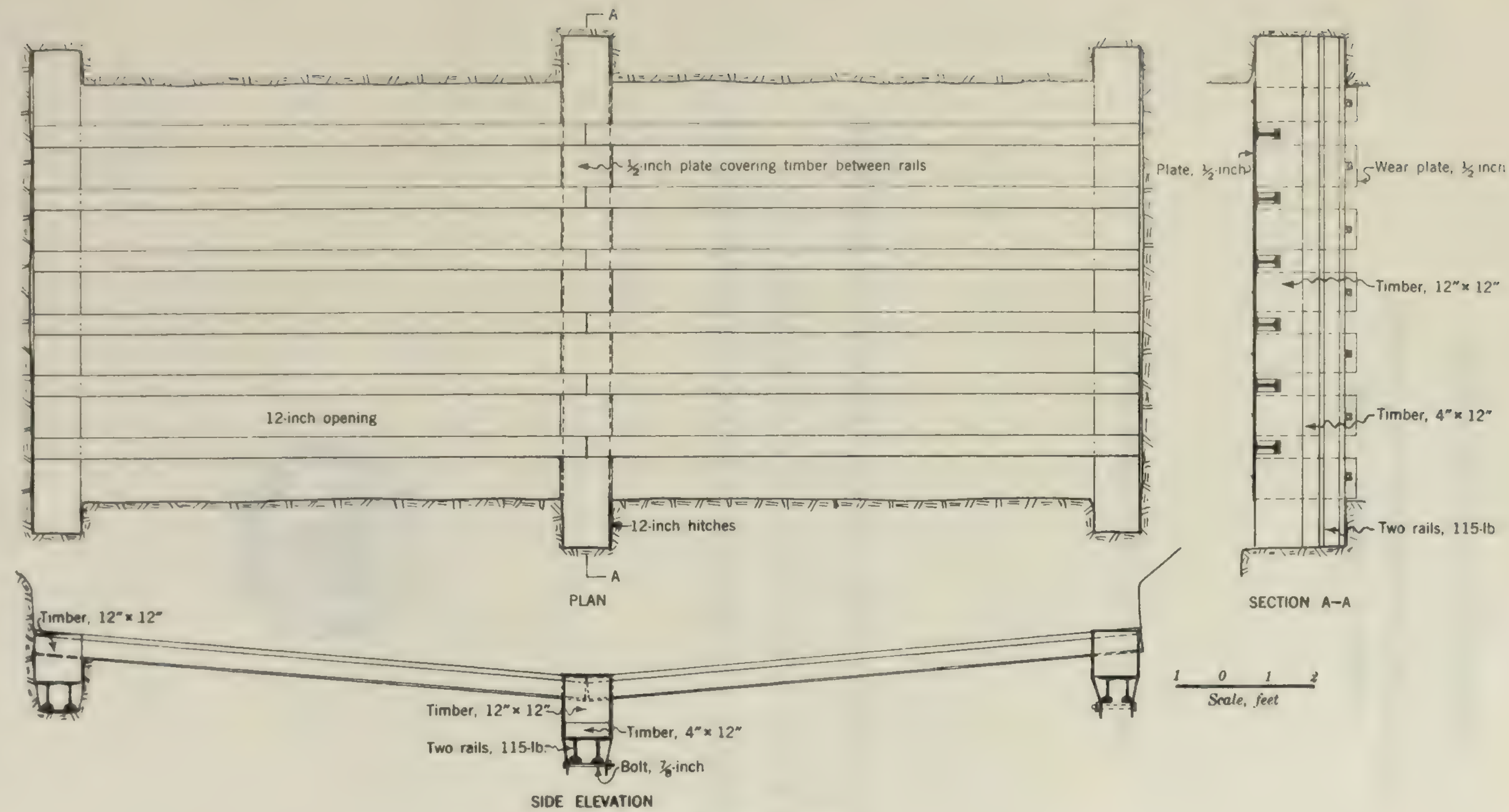

Figure 13.—Stope grizzly, 9 by 24 feet

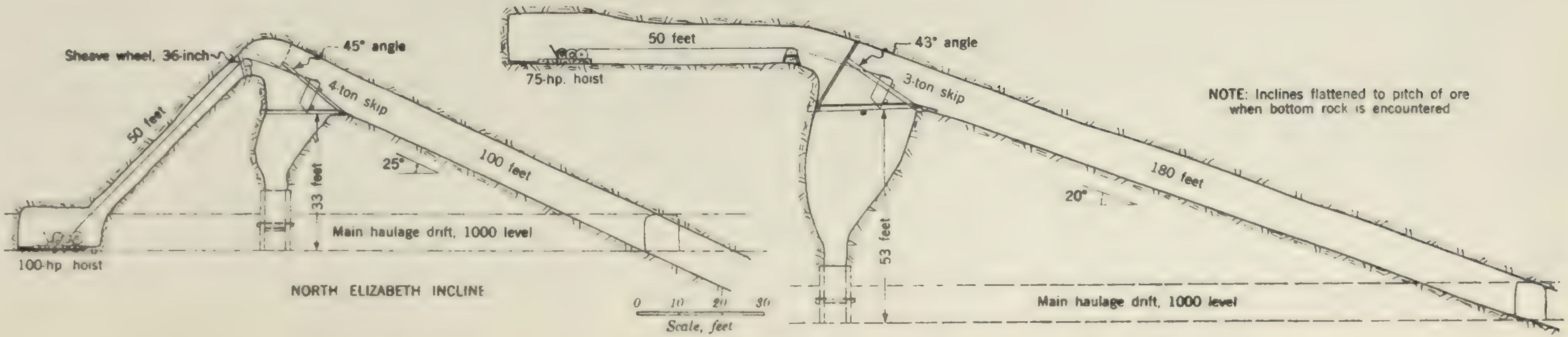

Figure 14.—Incline hoist stations and ore pockets

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SHAFT-SINKING METHODS, PRACTICES, AND COSTS OF THE CONSOLIDATION COAL CO. AT ITS NO. 261 MINE, CARETTA, McDOWELL COUNTY, W. VA.

BY

Laurence E. Kelley

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SHAFT SINKING METHODS, PRACTICES, AND COSTS OF THE CONSOLIDATION COAL CO. AT ITS NO. 261 MINE, CARETTA, MCDOWELL COUNTY, W. VA. [1]

By Laurence E. Kelley [2]

INTRODUCTION

This paper is one of a series being prepared by the United States Bureau of Mines on shaft-sinking methods and costs at individual operations in various mining districts of the United States. The methods and costs described herein are those of The Consolidation Coal Co. at one of its operations in the Tug River district, McDowell County, W. Va.

ACKNOWLEDGMENT

The author is indebted to Frank R. Lyon, vice-president, Thos. G. Fear, general manager of operations, and Chas. Enzian, chief engineer, of The Consolidation Coal Co., for valuable assistance and collaboration in the preparation of this paper. W. A. Cole, manager of the E. J. Longyear Development Co. furnished valuable data on methods and performance averages.

GENERAL DESCRIPTION OF PROJECT

The three shafts described herein were sunk by The Consolidation Coal Co. to develop a part of its Pocahontas coal lands. These properties lie about in the center of McDowell County, W. Va., and are underlaid with the No. 4 Pocahontas seam of coal, which is at a depth of from 500 to 600 feet below the surface. One of the shafts was for the purpose of ventilation and is known as the air shaft; another was to be used for handling materials and men into and out of the mine and is known as the manway shaft; the third of the shafts was for the purpose of hoisting coal and is known as the skip or main hoisting shaft.

A contract, covering the sinking and lining of all three shafts, was executed on August 15, 1922, and it was agreed that 15 months should be allowed for the completion of the air and manway shafts, and 24 months for

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6602."

2 - One of the consulting engineers, U. S. Bureau of Mines, and division engineer, Consolidation Coal Co.

the main hoisting shaft. The contractor was allowed a reasonable extension of time, however, in case of fire, strikes, and other things beyond his control. The E. J. Longyear Development Co., of Minnesota, secured the contract and completed the work. Figure 1 shows the topographical features at the shaft locations.

GEOLOGY

The geology of McDowell County, W. Va., has been thoroughly covered by Ray V. Hennen and Robert M. Gawthrop in a paper published by the West Virginia Geological Survey.

The shafts described herein were sunk through portions of both the New River and Pocahontas groups of the Pottsville Measures, which constitute the basal formation of the Pennsylvanian, the Allegheny Measures forming the upper portion. The New River strata are here about 1,300 feet thick, while the underlying Pocahontas strata, are about 720 feet thick. Each of the three shafts was sunk approximately 250 feet in the New River strata and 315 feet in the Pocahontas strata, total depth being 565 feet. The strata in this vicinity were deposited in the form of a great alluvial fan in Carbonaceous time. The center of this fan is at or near the town of Pocahontas in Virginia. The strata dip gently about N. 45° W.

Strata encountered in the sinking of these shafts consisted of numerous coal beds which were soft, multiple-bedded and columnar; lenticular sandstone with coal streaks; dark gray shale, argillaceous and silicious; dark, sandy, laminated shale; sandstone, massive and current bedded; dark, sandy shale and sandstone mixed; bluish gray, medium-grained sandstone; dark shale with abundant plant fossils; and finally the No. 4 Pocahontas coal, soft, multiple-bedded, columnar, excellent in quality, and 6 feet thick.

For a distance of about 300 feet below the surface the strata were water-bearing, which necessitated the application of grout to seal off the water. The surface materials, consisting of loose gravel, rock and clayey sand were fairly well consolidated, but showed a tendency to disintegrate on exposure to the atmosphere, especially in freezing weather. This made necessary the use of false formwork for about the first 30 feet of sinking. No well-defined fissures, slips, or fracture zones were encountered. Figure 2 shows a cross section of the formations through which the shafts were sunk.

SHAFT SINKING

(a) Preparation Period

In preparing for actual work of sinking, it was unnecessary to do a great deal of grubbing and clearing. The location had been formerly the site of a small farm in a narrow valley and the ground was already cleared of trees and brush. Strata were tested by diamond core drilling, five holes averaging about 600 feet deep were drilled at appropriate points over the location. An accurate topographic map was prepared showing contours on 1-foot intervals to be used for laying out temporary buildings as well as the permanent ones later. Prior to the drilling on the shaft locations, the entire property had been tested so that the quantity and quality of coal was known before the decision

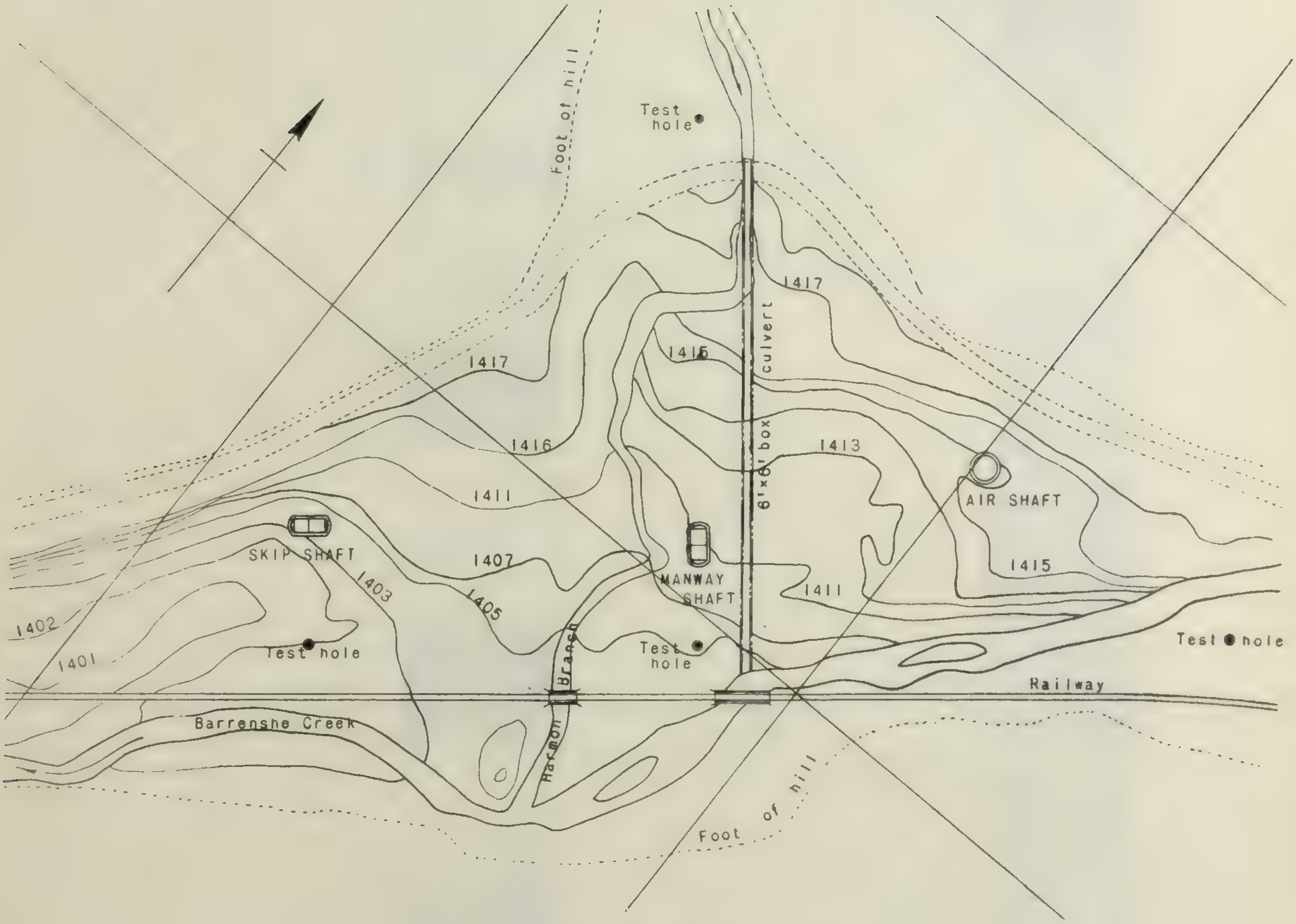

Figure 1.- Topography at shaft locations

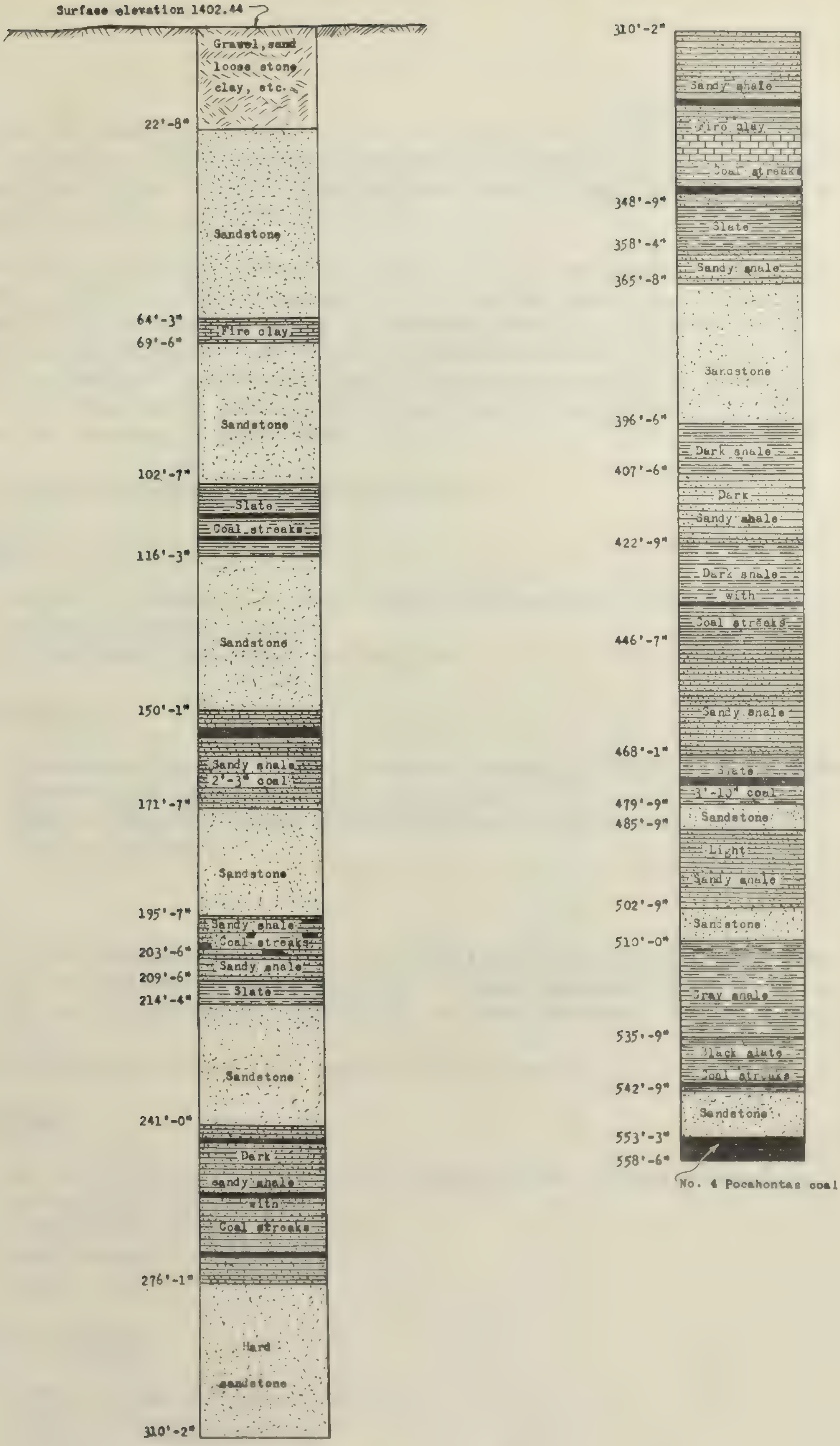

Figure 2.- Cross section of strata

was made to sink the shafts. A branch line of the Norfolk and Western Railway passed by the site so that it was necessary to construct only a siding to provide loading facilities for supplies and equipment. This siding was completed and ready for use on October 10, 1922.

Contractor was furnished a complete layout of the permanent buildings, pipe lines, etc., that the Coal Company would erect in order that his surface plant would be so placed as not to interfere with this construction. The contractor's surface plant consisted of three hoists and buildings, three headframes, boiler and air compressor house, cement storage house, steam heated bath house, mess hall, office building, shop building, and miscellaneous buildings for storage of materials.

A description of equipment used is as follows:

Power Plant

2 - 150-hp. 72 inch by 18 foot horizontal return tubular boilers.
1 - 60-hp. portable boiler.
1 - 20-kilowatt generator and engine.

Compressor Plant

1 - Class AA-2 Ingersoll-Rand 2-stage, straight-line, steam-driven compressor; capacity, 900 cubic feet.
1 - Class H Ingersoll-Rand duplex compound compressor; capacity, 840 cubic feet.

Hoists

1 - 12 by 14 inch Flory hoist S.D. 36 inch diameter, 30 inch face.
1 - 12 by 14 inch Flory hoist S.D. 48 inch diameter, 30 inch face.
1 - 12 by 18 inch Bacon hoist D.D. 60 inch diameter, 18 to 30 inch face.
1 - Brown hoist locomotive crane; 35-foot boom.

Shop Equipment

1 - No. 331 Ingersoll-Rand drill sharpener and threading machine; capacity, 6 inch pipe.
1 - Power drill press.
1 - Emery wheel.
Forge, etc.

Concreting Equipment

1 - Portable rock-crushing plant.
1 - Jeffry car unloader.
2 - 3/4-yard concrete mixers.
1 - Concrete blower.

Shaft Equipment

No. 50 and No. 55 Clipper rock drills.
1 - Ransome pneumatic grout mixer and placer.
Steam pumps.
26-cubic foot capacity (Missouri type) sinking buckets.
Wire rope.

Rock was hoisted out of shafts in buckets which were dumped into chutes emptying into side dump cars. Cars were pushed along a narrow-gauge track to the dumping point. Surface drainage was accomplished by means of a concrete culvert 6 by 6 feet in section. Figure 3 shows the arrangement of the surface layout and details of headframes.

(b) Shaft-Sinking Period

A plan and section of each of the three shafts are shown in Figure 4. Position of guides, buntons, etc., are shown as well as dimensions and depth of the shafts.

The main hoisting shaft was started on October 14, 1923, by means of the Brown Hoist locomotive crane. Muck was loaded directly into the 26 cubic foot bucket, attached to the hoist, and swung around and dumped near the top of the shaft. The following log will show the cycle of operations, number of men, and other pertinent data:

Date	Nature of work	Number of men	Man-hours
October 14, 1923	Mucking surface material	17.75	142
	Depth 3 feet		
15	Moving hoist	7	56
16	Moving hoist	6.25	50
17	Mucking	7.25	58
18	Mucking and timbering	6.50	52
19	Mucking-depth 6 feet	11	70
	Bailing water		18
20	Mucking, 25 buckets	9	30
	Timbering	9	42
21	Timbering	2	16
22	Mucking, 89 buckets	28.50	127
22	Timbering, depth 8 feet		8
23	Mucking, 131 buckets	24.75	190
	Timbering, depth 11 feet		8
24	Mucking, 28 buckets	12	40
	Timbering, depth 13 feet		56
25	Idle, crushing rock		
26	Timbering	1.25	10
27	Mucking, 59 buckets	9	70
28	Mucking, 58 buckets	17	84
	Timbering		38
	Piping, depth 17 feet	2	16

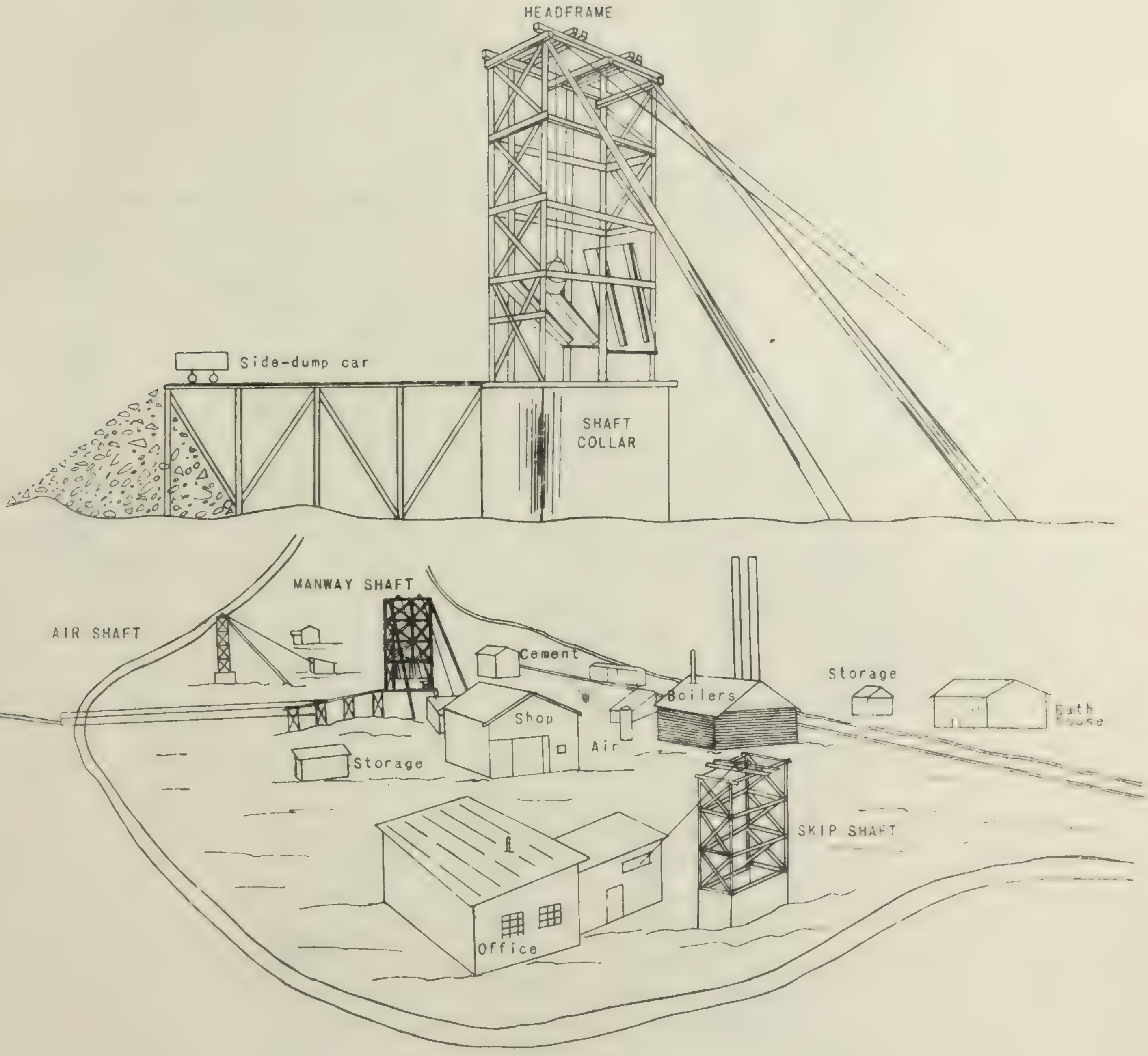

Figure 3.- Surface layout and headframe

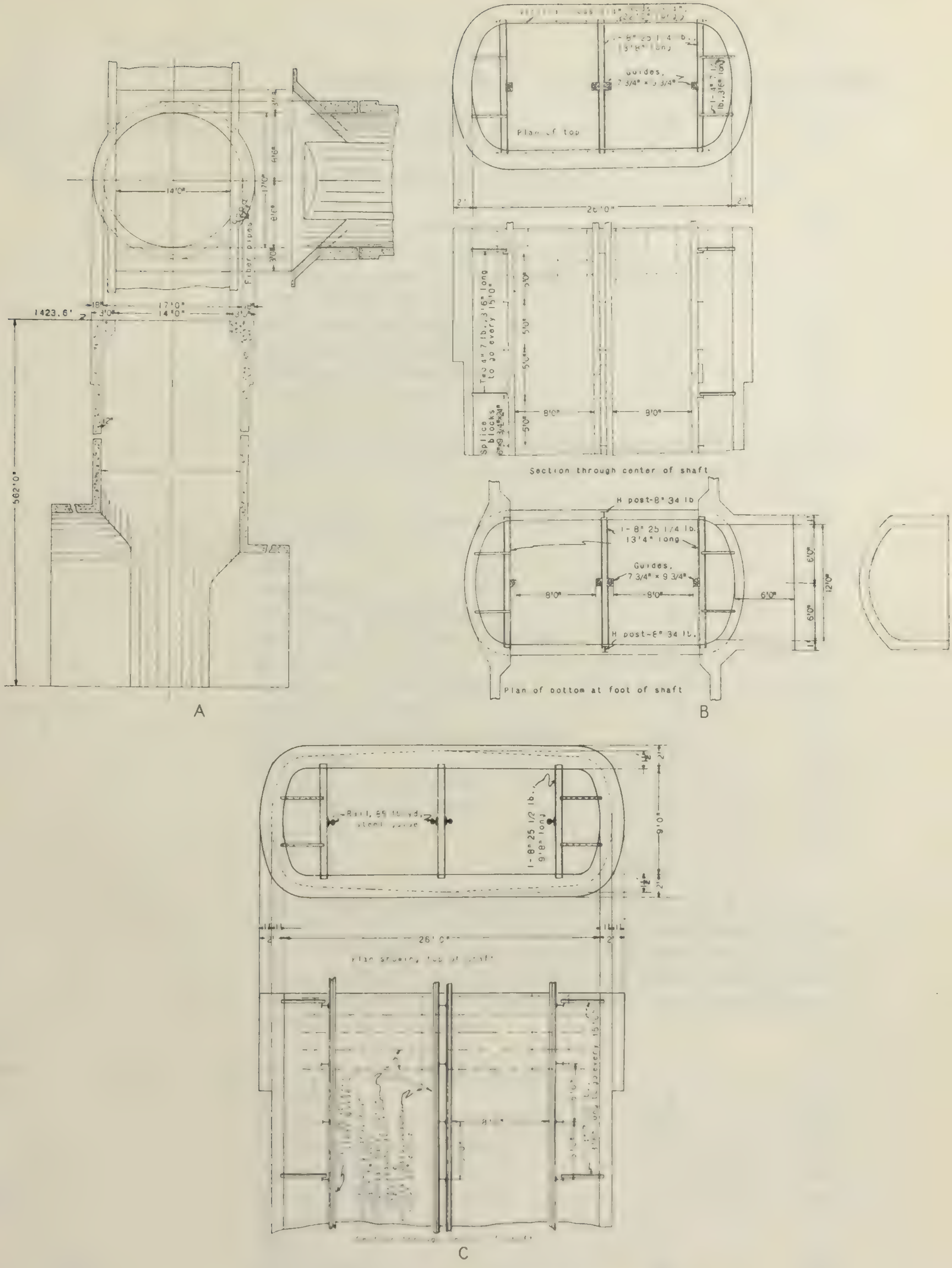

Figure 4.- Details of shafts: A, Air shaft; B, manway shaft; C, skip shaft

Date	Nature of work	Number of men	Man-hours
October 29, 1923 (Cont.)	Mucking, 4 buckets	3	6
	Timbering	3	18
30	Drilling test hole 20 feet deep	3	3
	Mucking, 21 buckets	3	21
	Depth, 18 feet; water struck in test hole going out 12 feet down		
31	Drilling test holes	3	2
	Mucking, 30 buckets	3	22
November 1	Mucking, 14 buckets	3	15
2	Timbering, depth 19 feet	3	9

At this depth all of the surface materials were removed and sinking through rock began as follows:

Date	Nature of work	Number of men	Man-hours
November 2, 1923	Drilling 20 holes, 126 feet	4	19
	Timbering	4	13
3	Idle		
4	Blasting	3	6
	Mucking, 23 buckets	3	16
5	Drilling 15 holes, 60 feet	3	3
	Blasting	3	4
	Mucking, 30 buckets	3	21
	Depth, 21 feet		
6	Drilling 14 holes, 52 feet	4	1
	Blasting	4	1/2
	Mucking, 42 buckets	4	27
7	Drilling 12 holes	4	4
	Blasting	1	1
	Mucking	4	27

The above procedure continued until at a depth of 33 feet, it became necessary to pump water at the rate of 5 gallons per minute for the full 24 hours. At a depth of 42 feet the first forms were placed for concrete lining and the work was then carried forward as follows:

Date	Nature of work	Number of men	Man-hours
November 27, 1923	Placing forms for lining (5 ft.)	4	44
28	Concreting, 5 feet, 198 bags	11	38
29	Placing forms, 5 feet	3	12
30	Concreting, 3 feet	7	56
December 1	Placing forms, 5 feet	7	6
1	Concreting, 2 feet	7	27
2	Idle		
3-4	Concreting	12	165
3-4	Placing forms	12	15

Date	Nature of work	Number of men	Man-hours
December 5, 1923 (Cont.)	Timbering	10	25
5	Concreting	10	54
6	Placing forms	13	15
	Concreting	13	69
7	Removing timbers (false formwork)	15	5

This work was continued until the 42 feet had been lined with concrete and permanent buntons installed, which work was completed on December 24. Three test holes for water were drilled on January 4, and it was found that water was coming into the shaft at the rate of 15 gallons per minute. One hundred and fifty bags of cement grout were forced into these holes, requiring 41 man-hours to complete the task. After using 272 additional bags of cement grout, the work of sinking was resumed. The shaft collar was completed on February 14, 1924, and by March 29 the regular hoist and headframe had been placed. The use of the Brown Hoist locomotive crane was discontinued at a depth of 58 feet.

From this point down, that is, after regular hoist and headframe had been put in service, the typical cycle of operations was as follows:

(1) Excavate from 50 to 60 feet of shaft.
(2) Place and align forms for concrete lining, 15 feet vertical.
(3) Concrete 15 vertical feet of shaft.
(4) Place and align 15 feet of forms.
(5) Concrete 15 vertical feet of shaft.
(6) Place and align 15 feet of forms.
(7) Concrete 15 vertical feet of shaft.
(8) Place and align 15 feet of forms.
(9) Concrete 15 feet of shaft.
(10) Place temporary guides.
(11) Extend piping and lower lodgment pump.
(12) Remove forms (60 feet).
(13) Place permanent buntons (60 feet).

The number of men per shift and per cycle necessary to complete 60 feet of shaft, together with the man-hours required for each class of work is as follows:

Class of work	Men per shift	Men per cycle	Man-hours per cycle	Number of shifts
Drilling	8	24	192	9.75
Blasting	8	24	192	5.90
Mucking	8	24	192	30.00
Placing forms	8	24	192	6.65
Concreting	8	24	192	6.20
Removing forms	8	24	192	3.46
Installing guides	8	24	192	2.70
Piping and pumps	8	24	192	2.00
Buntons	8	24	192	3.75

The above men made up the regular shift, and in addition to them there were a superintendent, an engineer, a shaft captain or foreman, and an assistant foreman. At times the work was carried on on a 2-shift basis. Mrn were transferred as needed on different operations. The number of shifts for the various classes of work times men per shift times eight hours will give the man-hours required to complete 60 feet of shaft as shown above.

DRILLING

Drills used in all three shafts were of the rotating hand-hammer type. These drills were used with 7/8-inch hexagonal hollow steels. They were of the dry type, and by the use of an air manifold it was possible to employ eight drills at once, requiring eight drill runners per shift or 24 men per day. The same men did the mucking. The chief advantage of unmounted drills lies in the fact that more flexibility can be had in the placing of holes to take advantage of the condition of the strata.

In the air shaft the pyramid-type cut was employed, the holes being so inclined as to form a sump at the center of the shaft. The average depth of the holes was about 6.4 feet, and gave about 6 feet of progress per round. The approximate average round is shown on Figure 5. Rounds varied somewhat with the hardness of the strata, the number of holes necessary in shale or slate being about one-third less than in hard or medium hard sandstone.

In the skip and manway shafts the "V" or wedge-shaped cut was employed. Two steeply inclined holes were drilled in the center of the shaft which when fired gave a sump in the middle of the shaft. The other holes were placed so as to take advantage of the face formed when the cut holes had been shot. The average depth of holes was about 6.4 feet.

An average of about 6 feet of drill steel was consumed in the drilling of a round, which made the cost of drill steel 27 cents per foot of shaft sunk. Usually three changes of steel were made in the drilling of a hole, the gages being 1 3/8, 1 1/4, and 1 1/8 inches, so that the holes at the bottom were never less than 1 1/8 inches in diameter.

Air for drilling was furnished by either a 900-cubic foot capacity, 2-stage, straight-line steam compressor or by an 840-cubic foot capacity, duplex compound compressor. Air pressure varied from 85 to 100 pounds per square inch.

BLASTING

A low-freezing, fumeless, 40 per cent gelatin dynamite, suitable for use under water, was used for all blasting. The cartridges were 1 1/8 by 8 inches. The detonator was placed in the first stick of powder to be inserted in the hole pointing toward the bulk of the charge. Delay electric blasting caps of No. 6 strength with 10 delays and 11 firing periods were used. The detonators were thoroughly waterproof and were provided with one white and one blue leg wire for use in parallel connections. Holes were well tamped and fired in the order shown on Figure 5.

About 1.3 pounds of powder were required per cubic yard and excavated. Ventilation after blasting was accomplished by use of air hose which was lowered and opened. Small blowers were also used, as well as the exhaust from the sinking pumps. Time required for drilling, as well as blasting and blowing, in man-hours per foot of shaft is as shown:

Man-hours per foot of shaft

Drilling	Blasting and Blowing
Man shaft - 13.7	Man shaft - 8.5
Skip shaft - 10.4	Skip shaft - 6.4
Air shaft - 11.6	Air shaft - 6.7

MUCKING

Mucking through surface material in each case was accomplished by means of a locomotive crane hoist and regular sinking bucket. Muck was loaded into the bucket and dumped near the top of the shaft by the crane. The average number of men mucking through the surface was nine, and they mucked 1.0 bucket in 1.32 man-hours. Surface material could be dug up by hand for a depth of about 20 feet in each shaft, after which it was necessary to drill and blast before the material could be loaded into buckets. An average of 5 men were employed per shift on this stage of the work; the average depth of holes drilled was 4.8 feet, and it required 0.59 man-hour per hole drilled. About 0.26 man-hour per hole was expended in blasting and blowing. The average round through the surface consisted of 18 holes, which required 42 hoists of one bucket each to remove. Total yardage excavated through surface in the manway shaft was 701 cubic yards requiring 1,383 man-hours for mucking, 118 man-hours for drilling, and 111 man-hours for blasting and blowing. In the air shaft these figures were: mucking 1,150; man-hours drilling 132; man-hours blasting and blowing 80, and total yardage excavated, 485.74. The figures for the skip shaft closely approximate those for the air shaft.

Mucking through the rock section of each shaft was carried forward in substantially the same manner. A description of the excavation ofthe manway shaft will be given and should show sufficient data, since the figures will apply to the other two shafts by taking into consideration their cross-sectional areas.

The excavation of the manway shaft required 252 1/2 days, including 21 2/3 days employed in excavating through the surface materials. Figure 3 shows surface arrangement and details of headframes used for sinking. Two buckets were used in each shaft in order to get balanced hoisting as well as to place buckets in convenient places for the muckers in the shafts. Buckets were dumped into chutes located in the headframes. Muck was passed from chutes into side-dump cars which pushed along a narrow-gauge track to the dumping point. The top of the collar was about 12 feet above the surface, which gave ample dumping space within a radius of 1,000 feet of the shafts. The number of men in the mucking or excavating crew was 24.1 per day, working on a 3-shift basis. Progress per day was 2.27 feet or 34.43 cubic yards of excavation. An average of 112.1 buckets was hoisted per day, the effective excavation per bucket being 8.3 cubic feet. In the sinking cycle 12.5 per cent of the time was

consumed in blasting and blowing, 20.3 per cent in drilling, 64.8 per cent in mucking, and 2.4 per cent was accounted for by delays on account of equipment failures.

Buckets were of 26-cubic foot capacity (Missouri type). Compressed-air drills were Nos. 50 and 55 Clipper rock drills, which with the hoists constituted the sinking equipment. The grouting machine was used at intervals to seal off water. The rock-disposal track was laid on temporary wood trestles and moved from time to time as necessity demanded. A feature of the top arrangement was the movable chutes which facilitated dumping of buckets as well as provided a measure of safety for the workmen below, since they almost completely closed the shaft while rock was being dumped into side-dump cars. Easy disposal of muck was accomplished by raising the shaft collars from 10 to 12 feet above the surface.

Total excavation in this shaft	yards	8,695
Effective excavation per man-hour . .	cubic feet	9.4

TIMBERING

Temporary timbering was necessary for a depth of 42 feet in the skip shaft through the surface, 41.73 feet in the air shaft, and 35.42 feet in the manway shaft. Wall plates were 8 by 10 inches and lagging was provided of 3-inch oak boards. Wall plates were placed at about 5-foot intervals. It required 15,960 board feet of timber for the skip shaft, 13,951 feet for the manway shaft, and 13,053 feet for the air shaft through the surface. In addition the air shaft required temporary guides and buntons all the way down, about 17,000 board feet of 6 by 8 inch timbers. Man-hours required for temporary timbering: skip shaft, 734; manway shaft, 504; and air shaft, 1,896. The wide variation between man-hours required for the air shaft and the other two was caused by the installation of temporary guides and buntons for its entire depth.

No permanent timbering was done in the air shaft. Both the skip and manway shafts have three rows of steel buntons made of 8-inch I-beams, 25 1/2-pound section. These were drilled and grouted in place in the concrete walls of the shafts after having been first thoroughly cleaned with steel brushes, painted with one field coat of graphite paint, and before being set in place covered with a second coat of heavy graphite paint. Recesses were left in the shaft walls when concrete was first poured, and after removal of forms the buntons were grouted in place. Time required for the work was 10.1 man-hours per bunton placed.

Permanent guides in the manway shaft are of 8 by 10 inch long leaf yellow pine dressed four sides to 7 3/4 by 9 3/4 inches with no length of less than 24 feet. The guides were sound, not bled, and painted with two good coats of hot carboline oil before installation. Time required for placing guides was 0.7 man-hour per foot of guides installed. Total footage of guides installed in this shaft was 2,264.

The permanent guides in the skip shaft were 85-pound steel rails. Four men were employed to place these, requiring 45 shifts or 1,440 man-hours,

equivalent to 0.6 man-hours per foot of guides installed. The total footage of guides installed in this shaft was 2,252.

The average number of men employed in setting buntons was about 5.3. For temporary timbering an average of 7.2 men were used. None of the timber was treated except as noted above. The following tabulation taken from a record of actual work done will show averages for temporary timbering:

Date	Number of men	Man-hours	Depth, feet
October 20	9	42	6
21	2	16	
22	9 1/2	8	8
23	8	8	11
24	12	56	13
26	1 2/8	10	15
28	8 1/2	38	17
	Average 7.2		

Man-hours per vertical foot timbered - 10.5

CONCRETING

All concrete work was done in accordance with the following specifications:

All concrete shall consist of one part Portland cement, two parts sand, and four parts crushed stone. Cement will be of a well-known brand, and will be purchased subject to the standard tests required by the American Society for Testing Materials, and must be ordered a sufficient time in advance to admit of testing, and shall be stored in a waterproof, steam-heated shed.

Sand shall be clean and sharp, free from all clay and loam, well screened, and of a quality to be approved by the engineer. The quality of the stone shall be subject to the approval of the engineer. It shall pass over a screen having openings not less than one-half inch apart and no piece shall exceed two and one-half inches in its greatest diameter.

Concrete is to be machine mixed, laid wet, and tamped against forms. The contractor shall keep men constantly working the mass against the forms while depositing concrete. Whenever concrete work is stopped overnight or for any length of time, before any new concrete is deposited, it shall be made thoroughly wet and grouted before concrete is started again."

A typical physical test on the cement used is as follows:

Initial set - more than . . . . minutes . 45
Final set - less than . . . . hours 10

Specific gravity - as rec'd	3.05
Specific gravity - ignited	3.16
Per cent on 200-mesh sieve	21.9
Tensile strength (7 days) . . . . pounds .	200

All shaft walls except those 24 inches thick were reinforced with 3/4-inch square twisted rod; the method used for this reinforcing is shown on Figure 4.

The stone aggregate was secured by means of a portable rock-crushing plant installed by the contractor. Concrete was mixed by 3/4-yard capacity mechanical mixers, dumped into sinking buckets, lowered into shaft, and poured behind forms, where it was thoroughly tamped and rammed.

Forms were constructed in 5-foot vertical sections to conform to shaft contour, and three sets were usually placed before pouring concrete. This gave 15 vertical feet of completed shaft. Details of forms used in each of the three shafts are shown in Figure 6.

In general about 60 feet of shaft was excavated and then concreted, the length of cycle being as follows:

Nature of work	Man-hours required		
	Skip shaft	Manway shaft	Air shaft
Placing forms	486	678	480
Pouring concrete	450	420	264
Removing forms-cleaning	252	390	240
Waiting for set	---	28	---
Miscellaneous delays	5	8	---
Man-hours per cubic yard specified	5.2	6.5	5.3
Man-hours per cubic yard poured	4.3	5.2	4.0

GROUTING

It was anticipated that large amounts of water would be encountered in the sinking of these shafts, and this anticipation was soon realized. Decision was made to apply cement grout both to make sinking easier and to make the shafts comparatively dry after they were completed. The wisdom of this procedure is now apparent since the shafts, after nearly seven years service, are sill fairly dry.

In the skip shaft the first grout was forced into the test holes at a depth of about 45 feet. Afterwards grouting was done through the concrete lining at a point only 19 feet below the surface. The final grouting in this shaft was done at a depth of 330 to 340 feet. The following summary shows some interesting data on this phase of the work in the skip shaft:

Section grouted, feet	Cubic yards excavated
19 to 41	301.9
41 to 176	1563.3
282 to 297	167.7
309 to 340	347.0
Total	2379.9

Total cost of grouting including test drilling and grouting through shaft lining was $13,945.51, which based on the above yardage figures, shows a cost of $5.86 per cubic yard excavated. This required a total of 1,046 1/4 barrels of cement; the cost per barrel was therefore $13.33.

Man-hours required per barrel of cement used for grouting was 5.16. The average depth of holes drilled for this purpose was 16.3 feet; the total number of holes drilled, 514; total footage drilled 8,398; 2,127 man-hours were required for the drilling. The man-shifts necessary required 85 bosses and 559 3/8 miners. The total number of shifts necessary to complete grouting in this shaft was 88.

In the air shaft it was necessary only to grout for a depth of 168 feet. The number of shifts required was 55. A total of 210 holes were grouted in this shaft, their average depth being 18.6 feet. A little over one-third as much cement was used in this shaft as was necessary in the skip shaft, the number of barrels used being 328.

The manway shaft required the largest amount of grout, a total of 1,278 1/4 barrels of cement being necessary, although the last grout was placed at a depth of 200 feet. A total of 369 holes were drilled for this purpose, their average depth being 14.9 feet. The man-shifts necessary required 85 bosses and 618 3/8 miners.

Grout was in all cases placed by means of pneumatic pressure, a Ransome pneumatic grout mixer and placer being employed for the purpose.

Table 1. - Personnel

	Skip shaft			Manway shaft			Air shaft		
	Number per shift	Number of shifts	Number per cycle	Number per shift	Number of shifts	Number per cycle	Number per shift	Number of shifts	Number per cycle
Underground:									
Supervision	1	3	3	1	3	3	1	3	3
Drilling - blasting	8	3	24	8	3	24	8	3	24
Mucking	8	3	24	8	3	24	8	3	24
Timbering	8	3	24	8	3	24	8	3	24
Concreting	8	3	24	8	3	24	8	3	24
Pumping	1	3	3	1	3	3	1	3	3
Surface:									
Hoisting	2	3	6	2	3	6	1	3	3
Landers - topmen	2	3	6	2	3	6	2	3	6
Timber framing	2	3	6	2	3	6	1	3	3
Repair shop	1	3	3	1	3	3	1	3	3
Compressormen	1	3	3	1	3	3	1	3	3
Concreting	2	3	6	2	3	6	2	3	6

The performance records show the number of miners and shift bosses employed. They carried on the work continuously and men were transferred as needed on different operations. In addition, there were a superintendent, an engineer, a shaft captain or foreman, and an assistant foreman. There were four or five surface laborers who were used on miscellaneous work such as handling materials. The repair shop for drill sharpening, etc., was manned by a master mechanic and two helpers.

Table 2. - Costs per foot of shaft

Item	Skip shaft			Manway shaft			Air shaft		
	Labor	Materials	Total	Labor	Materials	Total	Labor	Materials	Total
Excavation	$57.50	$50.74	$108.24	$74.00	$65.90	$139.90	$51.90	$45.57	$97.47
Concrete	25.30	37.00	62.30	27.10	42.88	69.98	16.30	23.70	40.00
Timbering - guides	1.34	2.53	3.87	1.93	2.19	4.12	---	---	---
Timbering - buntons	3.06	4.61	7.67	4.65	6.95	11.60	---	---	---
Grouting	18.80	5.80	24.60	20.29	6.91	27.20	11.19	1.82	13.01
Piping	---	---	---	0.25	0.45	0.70	1.00	1.80	2.80
Formwork - surface	0.20	0.13	0.33	0.16	0.11	0.27	0.08	0.06	0.14
Miscellaneous	---	---	---	2.17	1.46	3.63	1.68	1.12	2.80
Reinforcing steel	---	3.58	3.58	---	3.69	3.69	---	0.15	0.15
Totals	106.20	104.39	210.59	130.55	130.54	261.09	82.15	74.22	156.37
Sunk .... feet	-	-	563.00	-	-	571.58	-	-	562.53
Timbered do	-	-	563.00	-	-	584.08	-	-	---
Concreted do	-	-	559.40	-	-	584.08	-	-	572.13

Guides in manway shaft were 8 by 10 inch wood. Guides in skip shaft were 85-pound steel rails. All buntons were steel 8 and 4 inch I-beams.

Table 3. - Costs in units of labor, materials, and supplies

	Skip shaft	Manway shaft	Air shaft
A - Labor, man-hours per foot:			
1. Drilling and blasting	17.0	22.2	18.4
2. Mucking	32.1	43.8	32.5
3. Timbering - guides, buntons, etc.	5.4	10.2	3.4
4. Concreting	17.4	26.6	18.3
5. Hoisting	10.6	15.3	11.6
6. Rock disposal and miscellaneous surface	0.6	3.4	1.9
7. Pumping and piping	2.0	5.3	3.5
8. Miscellaneous labor including grouting	9.6	9.8	4.8
9. Supervision	10.6	15.3	11.6
Total labor per foot of shaft	105.3	151.9	106.0
Feet of shaft per man-shift	0.076	0.053	0.075
Average daily wage	$8.00	$8.00	$8.00
B - Supplies and power:			
1. Explosives - 40 per cent gelatin 1 1/8-inch sticks used per foot - - - - - - - - pounds -	15.3	19.8	13.7
2. Buntons used per foot - - - - - - - - pieces -	0.58	0.74	----
3. Guides per foot - - - - - - - - - linear feet -	4.0	4.0	----
4. Drill steel used per foot of shaft - feet -	0.8	1.0	0.7
5. Concrete per foot - - - - - - - cubic yards -	3.87	4.48	2.62

It will be noted that guides in the manway shaft were provided of wood, while those in the skip shaft were of steel - 85-pound steel rails. All buntons were of steel. In the manway shaft the buntons consisted of 2 pieces 20 inches in section, 348 pieces 8 inches in section, and 144 pieces 4 inches in section. Average time for installation was 10.1 man-hours per bunton placed. A total of 2,264 linear feet of guides was installed in the manway shaft, requiring an average of 0.7 man-hour per linear foot installed. The miscellaneous temporary timbering in the manway shaft required a total of 504 man-hours.

In the skip shaft the buntons consisted of 8 pieces 4 inches in section and 318 pieces 8 inches in section. Man-hours required per bunton placed was 7.2. Temporary timbering and placing of guides in this shaft required a total of 734 man-hours.

Table 4. - Performance Records and General Data

		Skip shaft	Manway shaft	Air shaft
Progress:				
Worked in shaft	days	253	364	280
Shaft completed per day	feet	2.22	1.56	2.01
Rounds completed per day (excavating time)		0.56	0.38	0.50
Progress per round	feet	6.0	6.0	6.0
Man-shifts, miners, and bosses		6020	8620	5971
man-shifts, timbermen		106	135	33
Drilling:				
Labor required	man-hours	5892	7885	6590
Drilled per round	holes	52	66	46
Drill steel used per round	feet	4.8	6.0	4.2
Drilled per man-hour	do	7.3	6.8	5.6
Drilled per foot advanced	do	78	98	69
Blasting and blowing:				
Labor required	man-hours	3644	4852	3797
Explosives used per foot of shaft	pounds	15.3	19.8	13.7
Mucking:				
Labor required	man-hours	18187	25003	18354
Effective excavation	cubic feet	178766	234768	160609
Effective excavation, per bucket	do	9.1	8.3	7.9
Effective excavation, per man-hour (mucking time)	do	9.8	9.4	8.8
Timbering:				
Labor required (temporary timbering)	man-hours	734	504	1896
Concreting:				
Placing forms and reinforcing	man-hours	4059	6524	4703
Pouring concrete	do	3753	4237	3077
Removing forms and cleaning up	do	2111	3639	2507
Concrete poured per foot of lining;				
In surface and collar section	cubic yards	6.37	9.1	4.22
In rock section	do	3.87	4.48	2.62
Total depth concreted	feet	559.4	584.1	572.1
Grouting:				
Drilling	man-hours	2127	1133	765
Placing grout	do	3276	4486	1947
Holes grouted	number	514	369	210
Depth of holes grouted, total	feet	8398	5501	3915
Cement placed	bags	4185	5113	1312
Shaft equipment:				
Installing guides	man-hours	1630	1587	----
Installing pipes	do	1260	3001	1958
Installing buntons	do	2318	3754	----
Miscellaneous work	do	361	1989	1047
Delays miscellaneous	do	488	1286	847

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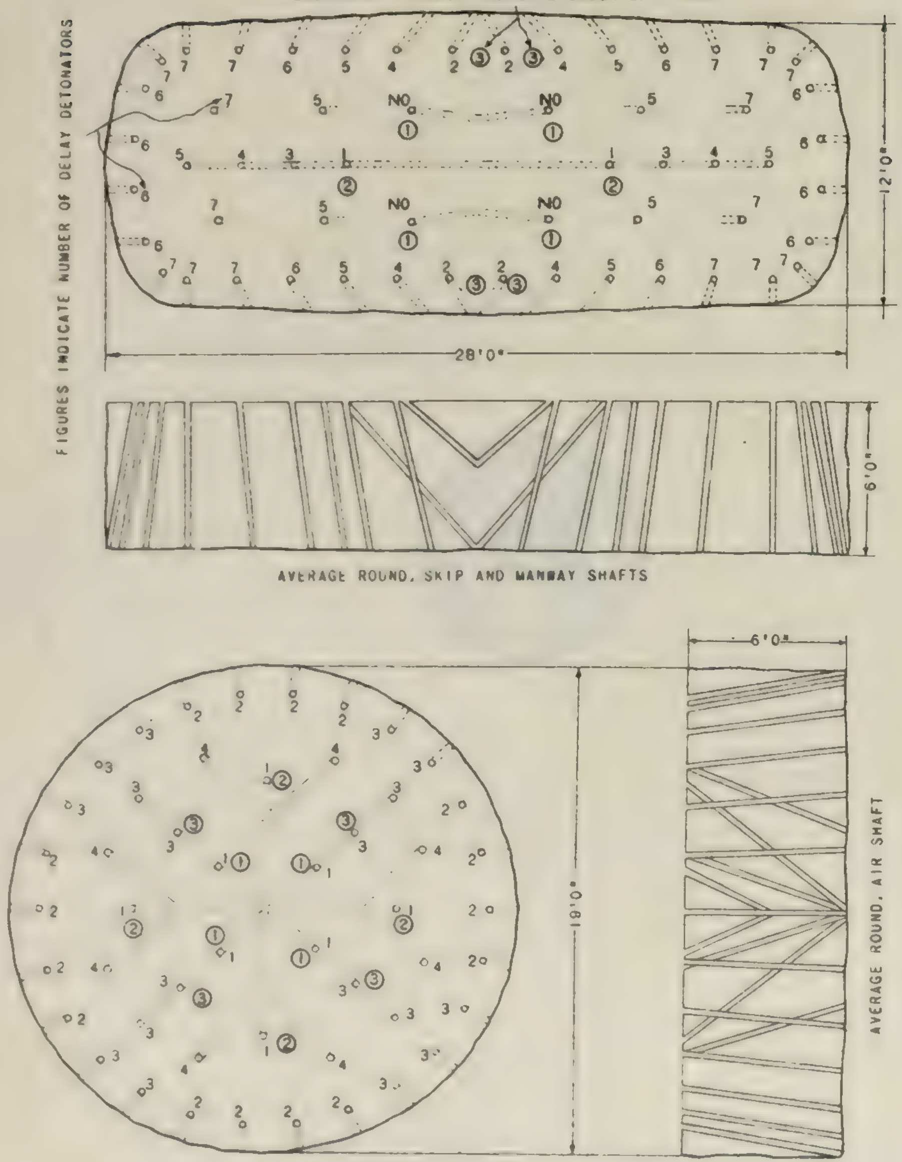

Figure 5.- Standard drill round and shooting method

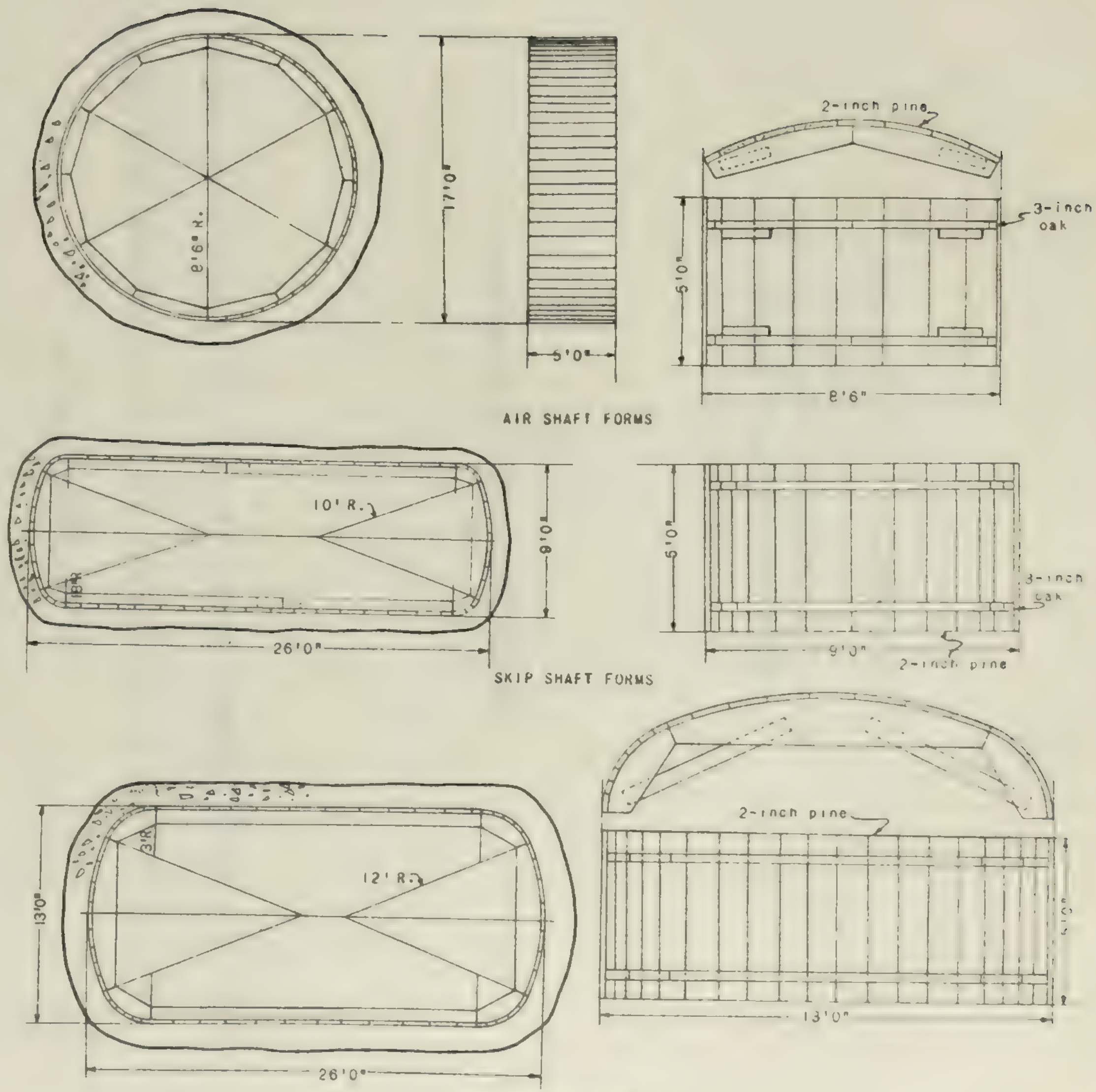

Figure 6.- Details of forms used in each of the three shafts

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

METHOD AND COST OF QUARRYING LIMESTONE AT THE MILLTOWN QUARRY OF THE LOUISVILLE CEMENT CO., MILLTOWN, IND.

BY

H. D. Baylor

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I.C.6603,
April, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHOD AND COST OF QUARRYING LIMESTONE AT THE MILLTOWN QUARRY OF THE LOUISVILLE CEMENT CO., MILLTOWN, Ind.[1]

By H. D. Baylor[2]

INTRODUCTION

This paper, describing the operations of the Louisville Cement Co. in quarrying limestone for the manufacture of quicklime, is one of a series being prepared for and published by the United States Bureau of Mines on stone quarries throughout the United States. These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total costs. It is recognized that publication of total costs might in many instances cause embarrassment to individual operators as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of the methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

ACKNOWLEDGMENTS

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HISTORY

The Louisville Cement Co. of Louisville, Ky., operates lime and commercial stone plants at Milltown, Ind., which is about 40 miles west of Louisville, Ky., on the Southern Railroad. The lime plant has a daily capacity of 200 tons of lump and pebble lime.

Two mixed-feed kilns were built during 1885, and quarry operations were started with hand labor. Later two additional kilns were built. A commercial-stone crushing plant was erected in 1901 to take care of the stone that covered the limerock. The mining of limerock was begun in the following year and was continued until 1902. This property comprises the No. 1 Plant (fig. 1).

Additional land was purchased during 1902, on which the No. 2 plant (figs. 1 and 2) was erected. The No. 2 plant consisted of two gas-fired vertical limekilns, a hydrated lime plant, a commercial stone plant, and a power house. In the same year the Eickel Stone and Lime Co. was purchased and called the No. 3 plant (figs. 1 and 2). It consisted of three mixed-feed kilns, a quarry, and a commercial-stone plant.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used. "Reprinted from U. S. Bureau of Mines Information Circular 6603."

2 - One of the consulting engineers, U. S. Bureau of Mines, and works manager, Louisville Cement Co.

When the No. 2 and No. 3 plants were put in operation, No. 1 was abandoned.

All rock for the No. 2 plant was taken from the open quarry by hand loading until 1918, at which time the overlying stone became so heavy that it was necessary to obtain the limerock from the No. 3 mines (figs. 1 and 2).

When the No. 2 commercial-stone plant was built, a quarry was opened adjacent to the plant. The material was handled with a railroad-type steam shovel.

During 1928 a rotary-kiln plant was erected and designated as No. 4 plant (fig. 1). The power house was abandoned in favor of central-station power. Two electric shovels with caterpillar trucks were purchased, one to replace the steam shovel and the other to be used in the lime quarry. At this time the mines were again abandoned and limerock was taken from the No. 2 quarry.

GEOLOGY

The stone in the Milltown quarries is of the lower carboniferous or Mississippian age and of the Mitchell formation. The Mitchell limestone is composed of compact limestones and chert, interbedded in places with beds of oolite and thin layers of limy shales. Its exposures occur over an irregular area, 3 to 25 miles in width, extending through the central part of southern Indiana, from the Ohio River in Harrison County, north and westward to the southwestern corner of Montgomery County, where they disappear beneath the glacial drift. The formation ranges from 250 to 350 feet in thickness at the quarry.

The Mitchell formation varies much in structural character and appearance. In most places it is a fine-grained crystalline or subcrystalline stone which is quite hard. Near the top and between the beds of grayish stone, layers having an oolitic structure appear nearly pure white in color and much softer than the gray stone (fig. 3). The chert occurs in varying quantities at different horizons in the gray stone. Often the chert nodules may increase in size and number from a few scattered pieces until they replace a greater part of the limestone. On weathered exposures the lime carbonate is bleached out and the chert fragments left in larger quantity than the residual limestone clay.

Both the oolite mentioned and the more common gray compact limestones (fig. 3) are used to manufacture lime. The following table gives the chemical analysis of each:

	SiO_2	R_2O_3	$CaCO_3$	$MgCO_3$
Kittle ledge	1.40	0.39	96.32	1.88
Oolite	0.87	0.20	96.77	2.16

The bedding is horizontal, with a slight dip to the southwest. The beds are comparatively thin and are traversed by many open joints. Accordingly, the stone can not be obtained in large pieces. These formations are typically cavernous and the ground surface bears the sink-hole type of topography.

Physical tests on the Mitchell limestone give the following data:

Specific gravity	2.66
Weight per cubic foot ... pounds	165.3
Water absorbed ... pounds per cubic foot	0.46
French coefficient of wear	11.6
Hardness	10
Toughness	8

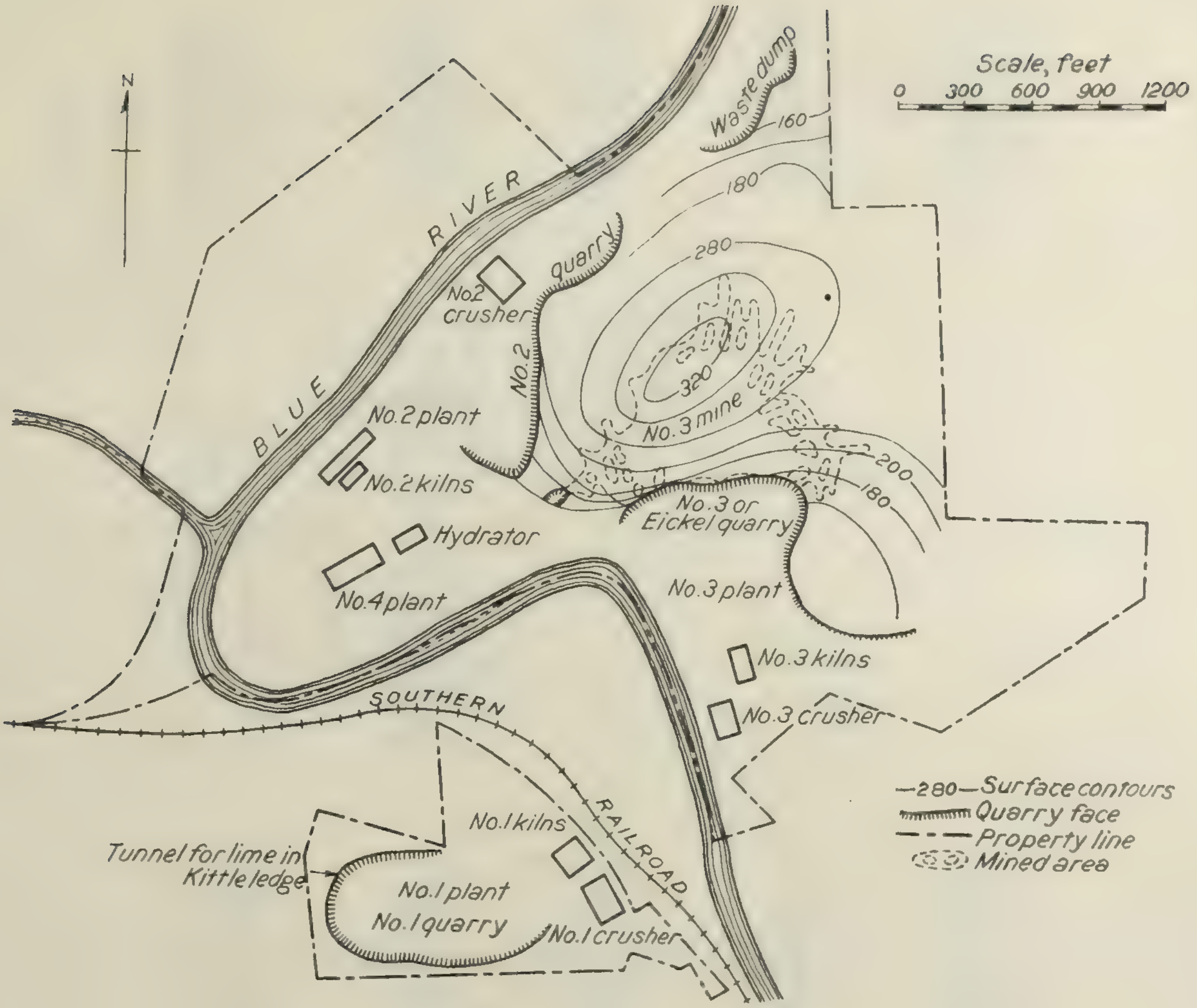

Figure 1.- Map of property

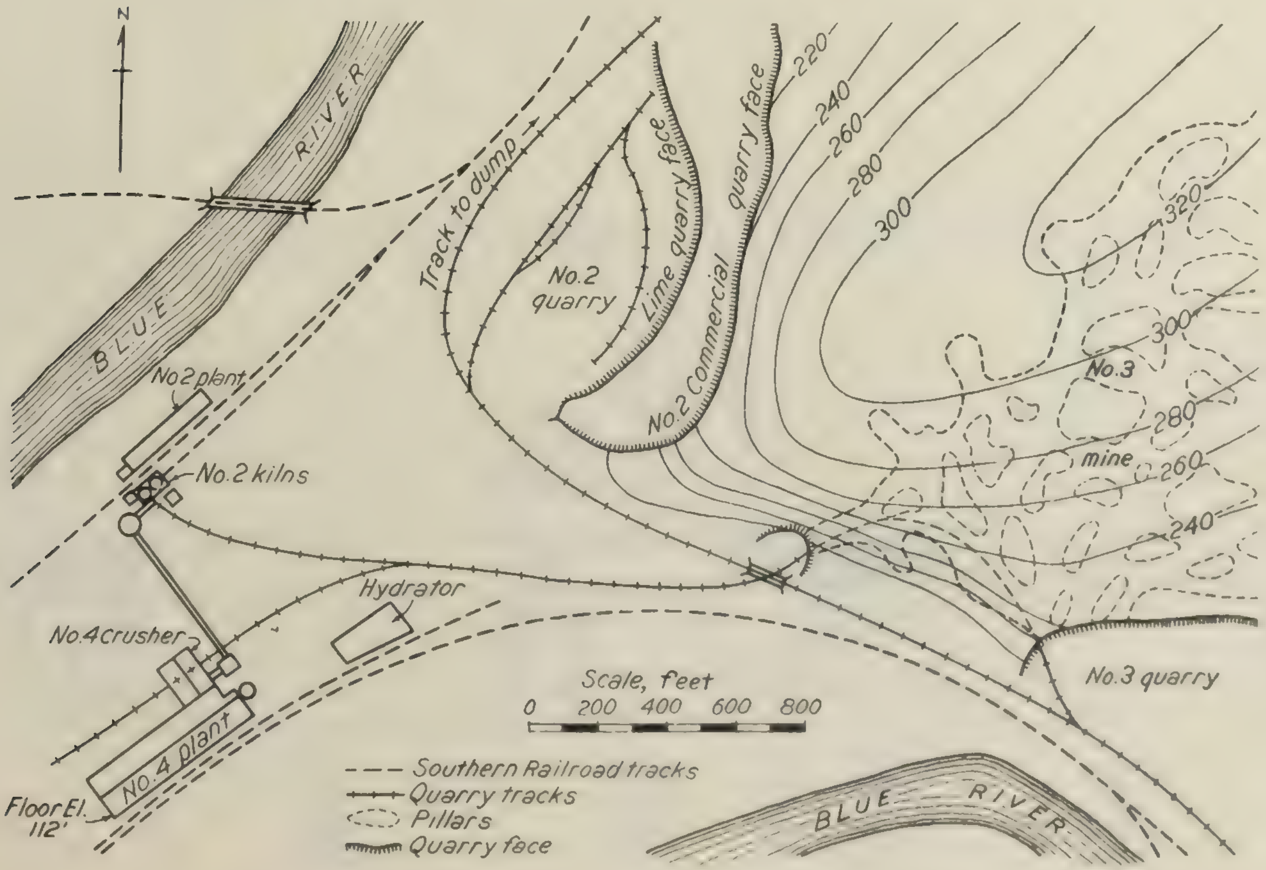

Figure 2.-Layout of quarries and plants

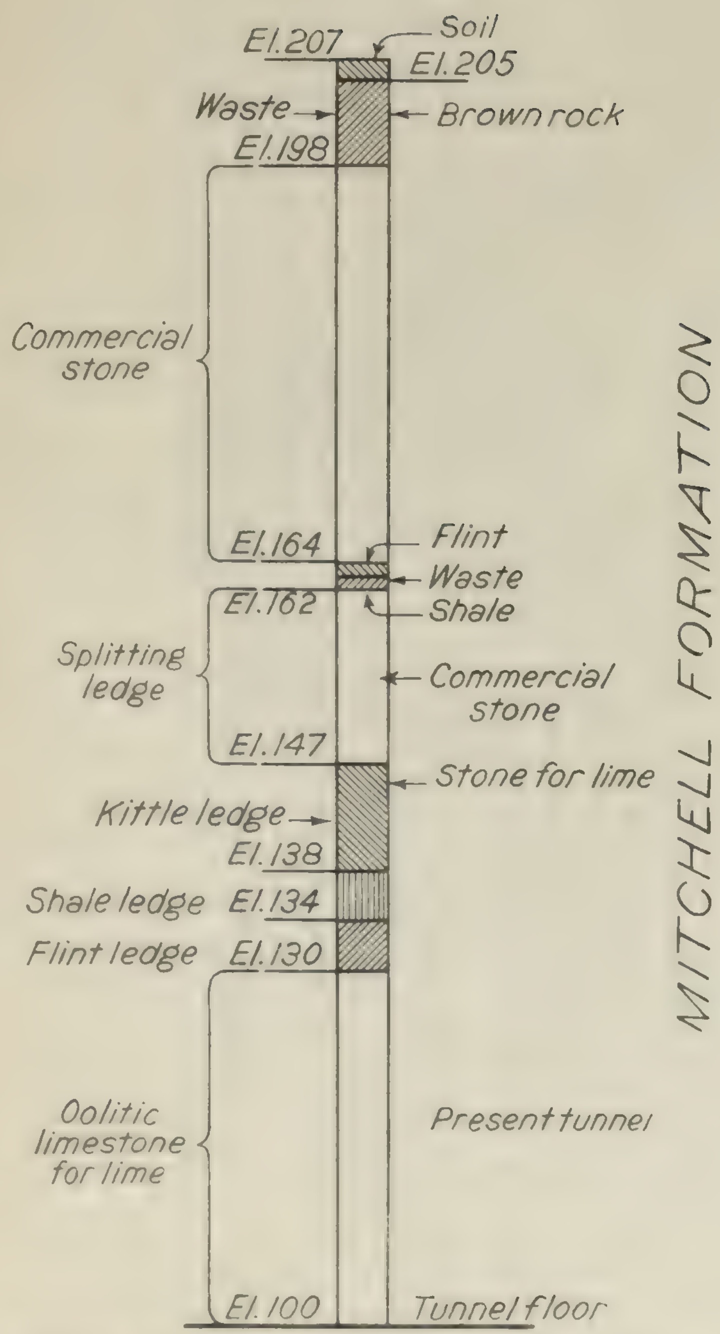

Figure 3.- Stratigraphic section of quarry

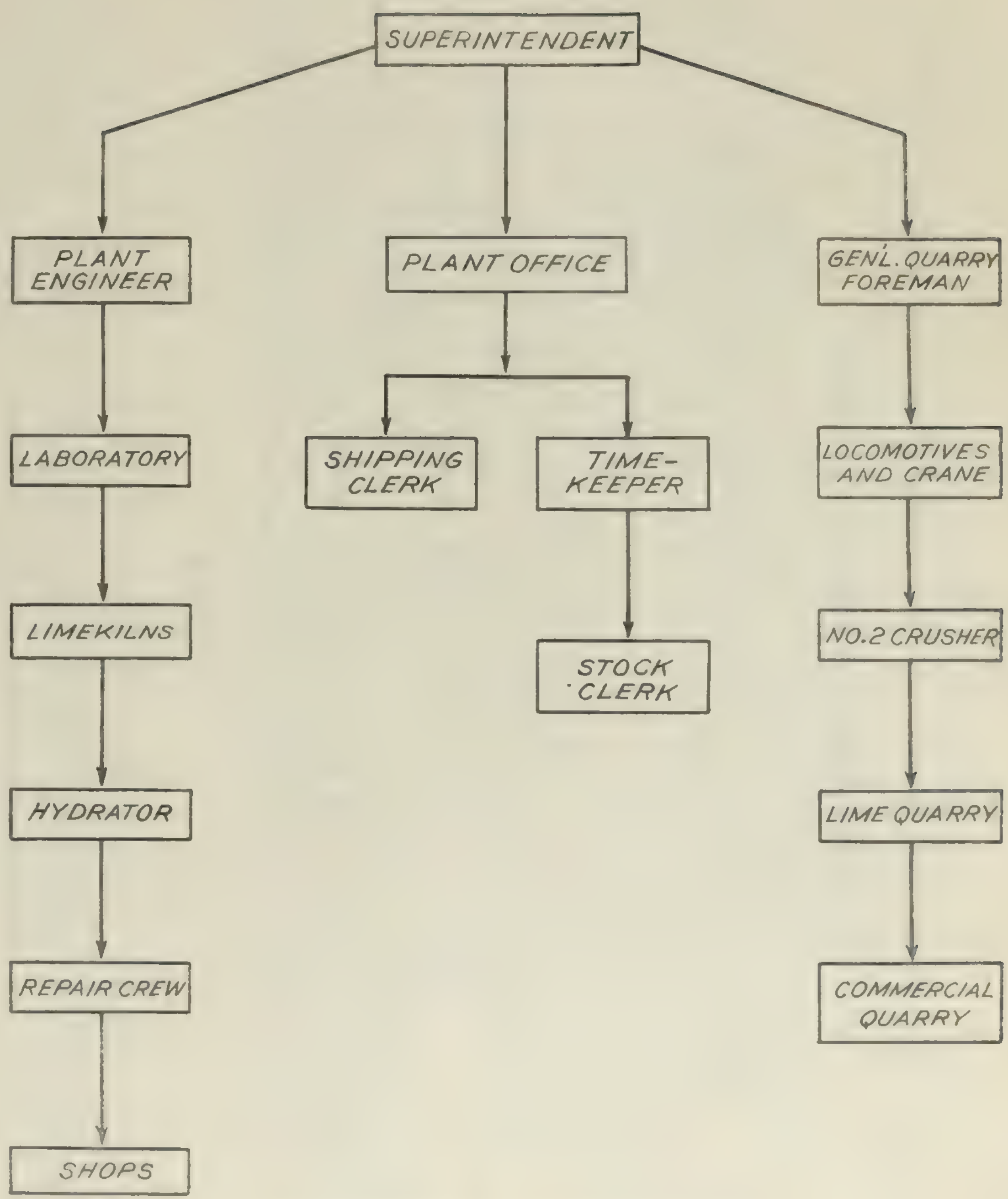

Figure 4.- Organization chart

The Kittle ledge (fig. 3) is the stone removed from the quarry described in this paper.

METHODS OF PROSPECTING AND EXPLORATION

Preliminary exploration in the area was accomplished by means of test pits and churn drills, but these means have been abandoned in favor of the diamond core drill. Test pits were found to be much more expensive than core drilling and were not as informative. In the area tested the limestone is covered by 2 to 7 feet of soil. Churn drills were found unsatisfactory in obtaining a knowledge of the character of the stone 75 feet to 100 feet below the top of the soil because with their use it was not possible to obtain sufficiently large pieces of the rock to ascertain its physical characteristics. In this instance also the churn drills available were the large, heavy type suited to drilling 6-inch blast holes. They were expensive to move. The core drill owned by the company was light and the two men who operated it could move it to any desired location with the aid of a team.

During 1928 considerable work was done with the core drill. The operating crew consisted of a driller and helper. Nine holes or a total of 564 feet were drilled at an average cost of $2.48 per foot for labor, $0.70 for supplies, and $0.57 for carbon or diamond wear, totalling $3.75 per foot. An accurate record of the chemical analyses of the stone and log of the hole was kept by the laboratory and is shown for two typical holes in Tables 1 and 2.

SAMPLING

No sampling is done in the quarry unless the complete lime analysis on the burned lime from the kiln show the $CaCO_3$ below an average of 96 per cent. From this lime analysis it is possible to tell whether the stone burned properly and whether its calcium carbonate content is up to the required standard. When this falls below standard the quarry is sampled by taking a grab sample from the rock broken ahead of the shovel.

CHOICE OF METHOD

The amount of overburden covering the limerock determines whether this operation will be an open quarry or mine. Figure 3 shows that the limerock is covered with 40 to 50 feet of commercial stone. The deciding factor between underground mining and operating the open quarry is a matter of cost. If the overburden can not be removed for less than 20 cents per ton of rock uncovered, mining is preferable. From this it can be seen that if the overburden is of such character as to necessitate drilling and blasting, a thick overburden would compel mining. A high market price for limerock or a thicker seam would enable a thicker overburden to be removed economically.

If the overburden can be sold, instead of wasted, as much may be removed from the lime deposit as can be disposed of at a profit or at such a loss as would not create a charge of over $0.20 per ton of limerock uncovered. When the sale of commercial stone is of sufficient volume to keep the lime ledges uncovered, the open-pit method prevails in the lime quarries; when it falls behind, the limerock must be mined.

All quarries at Milltown have a natural drainage. The rough topography of the land also lends itself well to building waste dumps.

Table 1.-Core Drill Analysis,
Milltown Commercial Stone Quarry, Drill Hole 2

Sample	Depth				Elevation		SiO_2	R_2O_3	$CaCO_3$	$MgCO_3$
	Feet	Inches	Feet	Inches	Feet	Inches				
1	35	10	36	8	162	5	5.76	0.70	90.61	2.93
2	36	8	38	4	160	9	4.24	0.60	93.94	1.22
3	38	4	40	0	159	1	4.32	0.36	93.16	2.16
4	40	0	41	8	157	5	3.82	0.98	93.62	1.58
5	41	8	43	4	155	9	4.58	0.96	92.83	1.63
6	43	4	45	0	154	1	5.20	1.74	91.28	1.78
7	45	0	46	8	152	5	2.42	0.76	95.17	1.65
8	46	8	48	4	150	9	2.78	1.58	80.73	14.91
9	48	4	50	0	149	1	3.82	1.30	93.28	1.60
10	50	0	51	8	147	5	9.80	1.42	86.73	2.05
11	51	8	53	4	145	9	7.94	1.94	74.07	16.05
12	53	4	55	0	144	1	0.30	0.32	97.46	1.92
13	55	0	56	8	142	5	1.32	0.68	96.08	1.92
14	56	8	58	4	140	9	1.18	0.64	96.24	1.94
15	58	4	60	0	139	1	1.38	0.44	96.68	1.50
16	60	0	65	0	134	1	2.14	0.44	91.77	5.65
17 T	65	0	65	10	133	3	2.24	0.70	94.06	3.00
17 B	65	10	66	8	132	5	5.90	1.04	76.34	16.72

Table 3.-Core Drill Analysis,
Milltown Commercial Stone Quarry, Drill Hole 3

Sample	Depth		Elevation		SiO_2	R_2O_3	$CaCO_3$	$MgCO_3$
	Feet	Inches	Feet	Inches				
1	96	8	163	4	4.82	0.24	92.96	1.98
2	98	4	161	8	3.82	0.22	93.70	2.26
3	100	0	160	0	1.96	0.38	95.61	2.05
4	101	8	158	4	2.90	0.24	94.46	2.32
5	103	4	156	8	5.66	0.48	93.10	0.57
6	105	0	155	0	2.40	0.24	95.12	2.24
7	106	8	153	4	2.00	0.28	95.51	2.21
8	108	4	151	8	4.80	0.34	92.50	2.32
9	110	0	150	0	5.34	0.82	72.88	20.96
10	111	8	148	4	6.80	0.32	90.91	1.97
11	113	4	146	8	8.84	1.40	65.51	24.25
12	115	0	145	0	1.26	0.52	95.94	2.28
13	116	8	143	4	1.58	0.34	96.49	1.59
14	118	4	141	8	1.36	0.22	96.52	1.90
15	120	0	140	0	1.40	0.48	96.36	1.76

QUARRYING METHODS

Stripping

No stripping is charged against the operation of the limerock quarry with which this report deals. The commercial-stone quarry removes stone which lies directly above the limerock quarry; hence, the operation of the commercial quarry is in one sense a stripping operation for the limerock. Overburden capping the commercial stone is removed, and the cost of such stripping is charged to that quarry.

There is some cleaning up to do on the limerock, but this is small in amount and no record is kept of it.

Drilling

The present lime quarry is worked in one 9-foot bench. Preliminary and secondary drilling is done with jackhammer drills using 1-inch hexagon steel and 6-point star bits which are sharpened by hand.

Holes are drilled approximately 10 feet deep, are 2 inches in diameter at the top and $1\frac{1}{4}$ inches at the bottom. One man does all the drilling. The average drilling speed is 17 feet per hour. Holes are spaced on 6-foot centers and 6 feet from the face of the ledge. Two lines of holes totalling from 20 to 30 are shot at one time.

Drilling costs are 12.33 cents per foot of hole, and 2.72 tons of rock are obtained per foot of hole.

Blasting

Forty per cent dynamite in $1\frac{1}{4}$ by 8 inch cartridges and No. 8 electric detonators are used. The detonators are joined to a common lead wire and shots are made by closing a switch on a 220-volt power line.

For secondary shooting, of which little is done, 40 per cent dynamite and No. 6 detonators with single-tape fuse are used.

The primary-blasting cost including labor, dynamite, etc., is 4.94 cents per ton. The secondary-blasting cost is 0.23 cents per ton. Ninety-six per cent of the dynamite used is for preliminary shooting and 4 per cent for secondary shooting. The total drilling and blasting cost is 9.90 cents per ton.

Loading

All stone is loaded with a 1-cubic yard, full-circle swing, electric shovel having caterpillar trucks. The shovel takes alternating current from the transmission line and converts it to direct current by a motor-generator set.

The loading crew consists of one shovel operator, and two quarrymen who pick impurities from the limerock. The daily output of the shovel is 270 tons. The shovel is capable of a greater capacity, but is delayed because impurities must be picked from the stone.

TRANSPORTATION

All rock is transported to the plant by means of one 14-ton steam Vulcan locomotive hauling trains of five cars each.

The cars used are of 5 cubic yard capacity, of steel construction, and are the 2-way rocker-dump type. When extra cars are needed, some 10-ton wooden cars are available. The track is 60-pound rail with a gage of 1 meter or 39.37 inches, and the average haul is 3,750 feet from quarry face to crusher. There is one grade in favor of the loads of 5 per cent for a length of 1,200 feet. Tracks are shifted by the regular quarry crew as needed, this cost being charged to railroad and switching.

CRUSHING PLANT

The crushing plant consists of a 48 by 36 inch jaw crusher driven at 19 r.p.m. by a 125-hp. motor through a Texrope drive. The average size of crusher feed is a 12-inch cube, and the discharge opening is 4½ inches. The crusher discharges into an inclined elevator having 42 by 18 by 24 inch buckets moving at a rate of about 60 feet per minute. The elevator is set at an inclination of 63° and is 63 feet from center to center. It is of double chain construction.

The elevator delivers to a 6 by 8 foot bar grizzly with 1¼-inch openings between bars on a slope of 40°. The rock passing over the bar grizzly goes to a 10-inch gyratory reduction crusher set for 1½-inch discharge. This crusher is driven at 400 r.p.m. by a 75-hp. motor through a Texrope drive.

Material passing the bar grizzly and reduction crusher drops to a vertical elevator using 24 by 12 by 18 inch buckets and operating at a speed of 85 feet per minute, and thence to a rotary screen. Rock slabs that will not pass a 1 5/8-inch opening are returned to the reduction crusher. Rock passing the rotary screen goes into a storage bin. The rotary screen is used only to remove the oversize rock for recrushing; no attempt is made at sizing other than this.

The screen is 33 inches in diameter by 7 feet long, is set on an inclination of 1-3/8-inches to 12 inches, revolves at 21 r.p.m., and is fitted with wire-screen cloth having 1-5/8-inch square openings.

The power consumed in the crushing plant varies considerably, ranging from 1 to 2.8 kilowatt-hours per ton of material crushed.

WAGE SCALE

All wages are paid on an hourly basis in accordance with the following rates:

	Wage per hour
Foreman...................	$0.52
Shovel engineer.........	.54
Powdermen................	.44
Driller and quarrymen	.39½
Locomotive engineer....	.44
Brakeman...................	.34½

SAFETY ORGANIZATION

All safety work is on a production basis, and each foreman is held responsible for the results obtained.

MANAGEMENT

Figure 4 is a chart showing the plant administrative organization. The lime-quarry foreman has charge of the drilling and blasting, loading of stone, and transportation to the plant. He receives his instructions from the general quarry foreman.

SUMMARY-OF COSTS

Table 3.-Cost per ton of limerock

Period covered: 1929 Tons produced: 47,187
Overburden: None

	Labor	Super-vision	Power	Explo-sives	Miscella-neous	Total
Drilling:						
Primary	0.0241	–	0.0212	–	–	0.0453
Secondary	.0011	–	.0009	–	–	.0020
Blasting:						
Primary	.0172	–	–	0.0322	–	.0494
Secondary	.0007	–	–	.0016	–	.0023
Loading	.0260	–	.0072	–	0.0047	.0379
Transportation	.0312	–	–	–	.0549	.0861
Miscellaneous	.0472	–	–	–	–	.0472
Depreciation	–	–	–	–	.0403	.0403
Insurance	–	–	–	–	.0031	.0031
Misc. overhead	–	0.0196	–	–	.0024	.0220
Total quarry cost	0.1475	0.0196	0.0293	0.0338	0.1054	0.3356

Table 4 -Lime-Quarry Costs in Units of Labor, Power, and Supplies

Period covered: 1929 Tons produced: 47,187

A. Labor:		
Drilling	man-hours per ton of stone	0.0406
Blasting	do.	.0406
Loading	do.	.0369
Haulage	do.	.0812
Miscellaneous	do.	.1069
Supervision	do.	.0378
Total labor	do.	0.3440
Average tons per man per shift		29.06
Labor per cent of quarry cost		49.6
B. Power supplies:		
Explosives	pounds per ton of stone	0.256
Total power	kilowatt-hours per ton of stone	2.05
1. Shovel	do.	0.50
2. Drills	do.	1.55

Table 5.-Lime-Quarry Average Cost, Direct Operation

Period covered: 1929 Tons produced: 47,187
Type of shovel: Marion Electric Type 7, Caterpillar trucks
Size dipper: 1 Cubic Yard

	Cost per ton
Engineers	$0.0221
Total operating labor	.0221
Grease and lubricants	.0009
Power	.0072
Total supplies	.0081
Shovel and shop repair labor	.0039
Repair supplies	.0038
Total repairs	.0077
Total shovel operation	0.0379

Table 6.-Summary of Lime-Quarry Costs

Period covered: 1929 Tons produced: 47,187

	Cost per ton
Loading	$0.0379
Drilling:	
Operating labor	.0213
Air compressor, power	.0221
Repair labor	.0039
Total drilling	.0473
Blasting:	
Labor	.0179
Explosives	.0338
Total blasting	.0517
Haulage:	
Locomotive labor	.0312
Fuel and other supplies	.0549
Total haulage	.0861
General charges	.1126
Total cost per ton	0.3356

Table 7. Cost Per Ton of Quarrying Limerock

Period covered: 1929 Tons produced: 47,187

	Cost per ton
Operating labor	$0.0078
Repair labor	.0009
Operating supplies	.0038
Explosives	.0338
Power	.0293
Railroad and switching	.0861
Teaming	.0024
Depreciation	.0403
Insurance	.0031
Total cost	0.3356

Table 8. Explosives Used in Quarrying Limerock

Production..........tons..........47,187.0
Explosives..........pounds..........12,099.4

Ratio: 1 pound of explosive gives 3.9 tons stone.
Ratio: Pounds of explosive per ton of stone - 0.256.

Table 7.-Cost Per Ton of Quarrying Limerock

Period covered: 1929 Tons produced: 47,187

	Cost per ton
Operating labor	$0.0078
Repair labor	.0009
Operating supplies	.0038
Explosives	.0338
Power	.0293
Railroad and switching	.0861
Teaming	.0024
Depreciation	.0403
Insurance	.0031
Total cost	0.3356

Table 8.-Explosives Used in Quarrying Limerock

Production	tons	47,187.0
Explosives	pounds	12,099.4

Ratio: 1 pound of explosive gives 3.9 tons stone.
Ratio: Pounds of explosive per ton of stone - 0.256.

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April, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

METHODS AND COSTS OF CONCENTRATING SCHEELITE ORE AT THE SILVER DIKE MILL, MINERAL COUNTY, NEVADA

BY

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHODS AND COSTS OF CONCENTRATING SCHEELITE ORE AT THE SILVER DIKE MILL, MINERAL COUNTY, NEV.[1]

By William O. Vanderburg[2]

INTRODUCTION

This paper describing the methods and costs of milling scheelite ore at the Silver Dike concentrator of the Nevada-Massachusetts Co., Inc., is one of a series being prepared by the Bureau of Mines.

In the Silver Dike mill tungsten ore which contains about 1 per cent of scheelite is treated at the maximum rate of 45 tons per 24 hours by tabling followed by magnetic separators to clean the table concentrates. Unusually high-grade concentrates are produced which average between 70 and 75 per cent of WO_3.

ACKNOWLEDGMENT

The author acknowledges the assistance rendered in the preparation of this paper by Ott F. Heizer, general manager of the Nevada-Massachusetts Co., Inc., and W. G. Emminger, superintendent of the Silver Dike subsidiary of the Nevada-Massachusetts Co., Inc.

LOCATION

The Silver Dike mine and concentrator are on the east flank of the Excelsior Mountains, 12 miles by automobile road southwest of Mina, Mineral County, Nev. The locality is known generally as the Silver Star or Gold Range mining districts. The Hazen branch of the Southern Pacific Railroad connects Mina, the southern terminus of the branch, with the main trunk line at Hazen; trains between Mina and Hazen are run on a daily schedule. The Tonopah & Goldfield Railroad operates a train daily between Mina and Goldfield via Tonopah. A narrow-gage railroad also connects Mina with southern California points by way of Belleville.

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1 - The Bureau of Mines will welcome reprinting of this paper providing the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6604."

2 - Associate mining engineer, U. S. Bureau of Mines.

The Silver Dike property is accessible by automobile road from Mina and all supplies and concentrates are hauled over this road by truck. The topography in the immediate vicinity of the mine and mill is of high relief; the climate is semiarid. The mill is built on the side of a steeply walled canyon. This site presents two features which adversely affect milling operations: namely, insufficient space for the disposal of tailings and a scant water supply.

The elevation of the upper end of the mill building is 6,470 feet above sea level.

POWER

Power for the mine and the mill is generated at the Mono Lake hydro-electric plants of the California-Nevada Power Co. at Bishop Creek and is purchased from the Mineral County Light and Power Co. under a schedule which varies from $0.025 to $0.04 per kilowatt-hour, depending upon the amount used. There is a minimum monthly demand charge in the schedule. The average cost of power when producing scheelite at two-thirds capacity is approximately $0.03 per kilowatt-hour.

From Bishop Creek the power is transmitted at 55,000 volts to a point 6 miles distant from the mine where delivery is accepted. At this point it is stepped down to 6,600 volts for transmission to the mine and mill. At the property it is stepped down further to 440 volts for use by all motors.

The mine is provided with three 30-KV-A. transformers, the mine crushing plant has two 20-KV-A. and the mill three 100-KV-A. transformers. All transformers are equipped with Pellet oxide-film lightning arresters.

Power for emergency lighting purposes is furnished by a 1,500-watt Kohler plant which consumes 1 gallon of gasoline for each four hours that the unit is operating.

WATER

Water for milling and domestic purposes is obtained from two wells sunk in the canyon, one above and the other below the mill site. The lower well which furnishes water for the mill is 50 feet deep and has a flow of 6 gallons per minute. The water is pumped from this well to a 1,200-gallon capacity tank by a No. 6 Stover deep-well pump; from this tank it is elevated about 125 feet to the mill supply tank by a 3 1/2 by 6 inch Fairbanks-Morse horizontal duplex pump. Water for domestic use is supplied by the upper well which is 35 feet deep. It is pumped from the well intermittently and flows by gravity through a 1-inch pipe line to the camp.

The amount of water available in proximity to the property during comparatively dry periods is insufficient for full capacity milling operations and the mill operation is limited to two shifts per day for this reason. A water right in Spearmint Canyon in the Pilot Mountains 12 miles distant

has been acquired to overcome this difficulty, and eventually the deficiency in the supply of mill water will be made up from this source.

Equipment has been provided for reclaiming a portion of the water contained in the tailings pulp; the water so recovered is approximately 75 per cent of the mill water used.

ORE TREATED

The ore occurs in quartz veins which have an average dip of about 75° and an average width of 3 feet. The length of the ore shoots varies from 60 to 120 feet. The width of the vein mined is determined more by the scheelite content than by structural characteristics of the vein. The country rock is monzonite, and a number of inclusions of this rock are found in the vein filling.

Practically all the past production of scheelite concentrates in the Great Basin region has been derived from contact-metamorphic deposits consisting of limestones altered to tactite by solutions accompanying intrusions of granitic rocks. The Silver Dike deposits are quartz veins of the replacement type and constitute an exception to the usual occurrence of scheelite ore in this district. The ore, in consequence, does not contain serious amounts of copper, bismuth, molybdenum, phosphorous, or other undesirable elements ordinarily present in minerals associated with the contact-metamorphic deposits

The scheelite crystals in the Silver Dike ore vary in size from 1/16 inch to 2 inches in diameter. Pyrite and, to a lesser extent, manganese sulphide are present in the ore.

The ore is mined by a system of shrinkage stopes; the top of the broken ore in the stopes is kept at a suitable distance from the back by drawing off the excess ore through plank chutes into cars. The ground is solid and little timber support is required. In blasting, practically all of the ore is broken to such size that it will pass through a grizzly with 8-inch openings. Entry to the mine is made by an adit crosscut 900 feet long. Ore is transported to the mine crushing plant by a locomotive operated by storage batteries; the average length of haul is 1,360 feet underground plus 300 feet from the portal of the adit to the bin.

Hand sorting of ore and waste is not practiced at Silver Dike, as it is difficult to distinguish ore from waste because of the intimate association of the mixture and the similarity in appearance of scheelite and quartz. An effort is made to maintain the grade of the ore broken at 1 per cent of scheeli which is equivalent to 0.8 per cent of WO_3, but close supervision in mining and numerous pannings have been found necessary to prevent the admixture of ore and waste in mining operations.

HISTORY OF CONCENTRATOR OPERATIONS

The present Silver Dike property represents a consolidation of two properties originally known as the Silver Dike and Wagner properties. The consolidation comprises a group of 12 contiguous claims. The properties were located for gold and silver in 1915. Although the occurrence of tungsten was known at the time of discovery no effort was made to develop the properties for tungsten until the extraordinary demand for this metal developed during the World War. At this time the Silver Dike property was owned and operated by the Atkins-Kroll Co. of San Francisco. The ore was hauled from the mine to a mill at Sodaville, a distance of 8 miles, by wagons and tractors. The Sodaville mill had capacity to treat 50 tons of ore per day.

In November, 1918, the mine closed down, and shortly afterwards the Sodaville concentrator was dismantled, due to almost complete paralysis of the tungsten industry in the United States. From 1919 to 1924 the production of tungsten concentrates in Nevada ceased because of the depressed condition of the industry.

A 25-ton capacity mill was built on the Wagner Co. property by lessees in 1926 and was operated by them until 1927. The method of treatment used consisted of crushing to 10-mesh size by a jaw crusher and rolls and concentrating the crushed product by tables. The tables produced finished concentrates.

In 1929 both the Wagner and Silver Dike properties were acquired by the Nevada-Massachusetts Co., Inc., under bond and lease. The present mill was erected in 1930.

PRESENT METHOD OF CONCENTRATING

A summary of the present method of ore treatment follows:

1. Crushing mine ore to 3/4-inch size by two Blake-type crushers which operate in series.

2. Crushing from 3/4-inch to minus 12-mesh by two sets of rolls operating in series; the primary rolls are in closed circuit with a trommel having 7/16-inch holes, and the secondary rolls with a 12-mesh Callow screen.

3. The minus 12-mesh material is sized by a 22-mesh Callow screen. The oversize is treated on an Overstrom table; the undersize pulp after thickening is also treated on an Overstrom table. The tables produce concentrates, middlings, and waste tailings.

4. The combined table concentrates, after drying, are given a short roast to convert pyrite to a magnetic iron sulphide; the sulphide is then removed by a Dings magnetic separator.

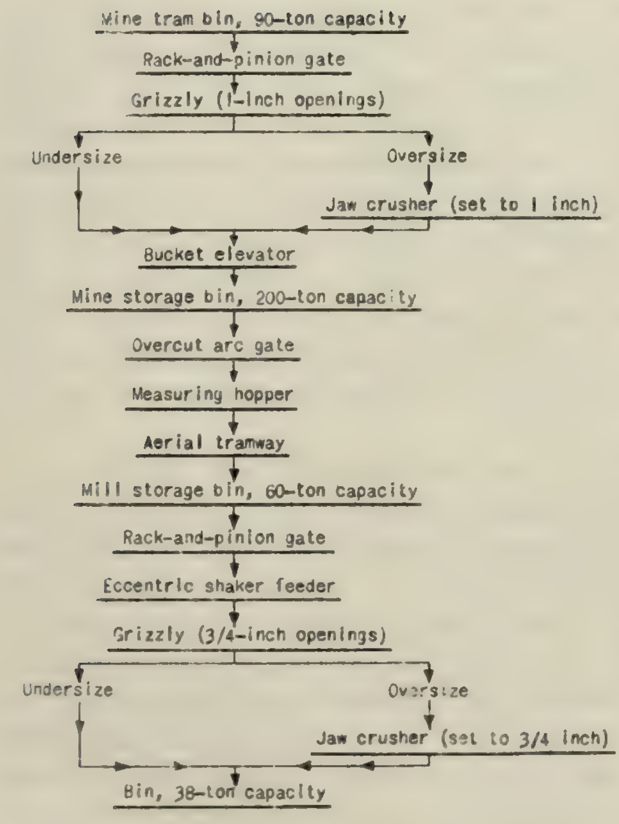

Figure 1.- Flow sheet of coarse and intermediate crushing

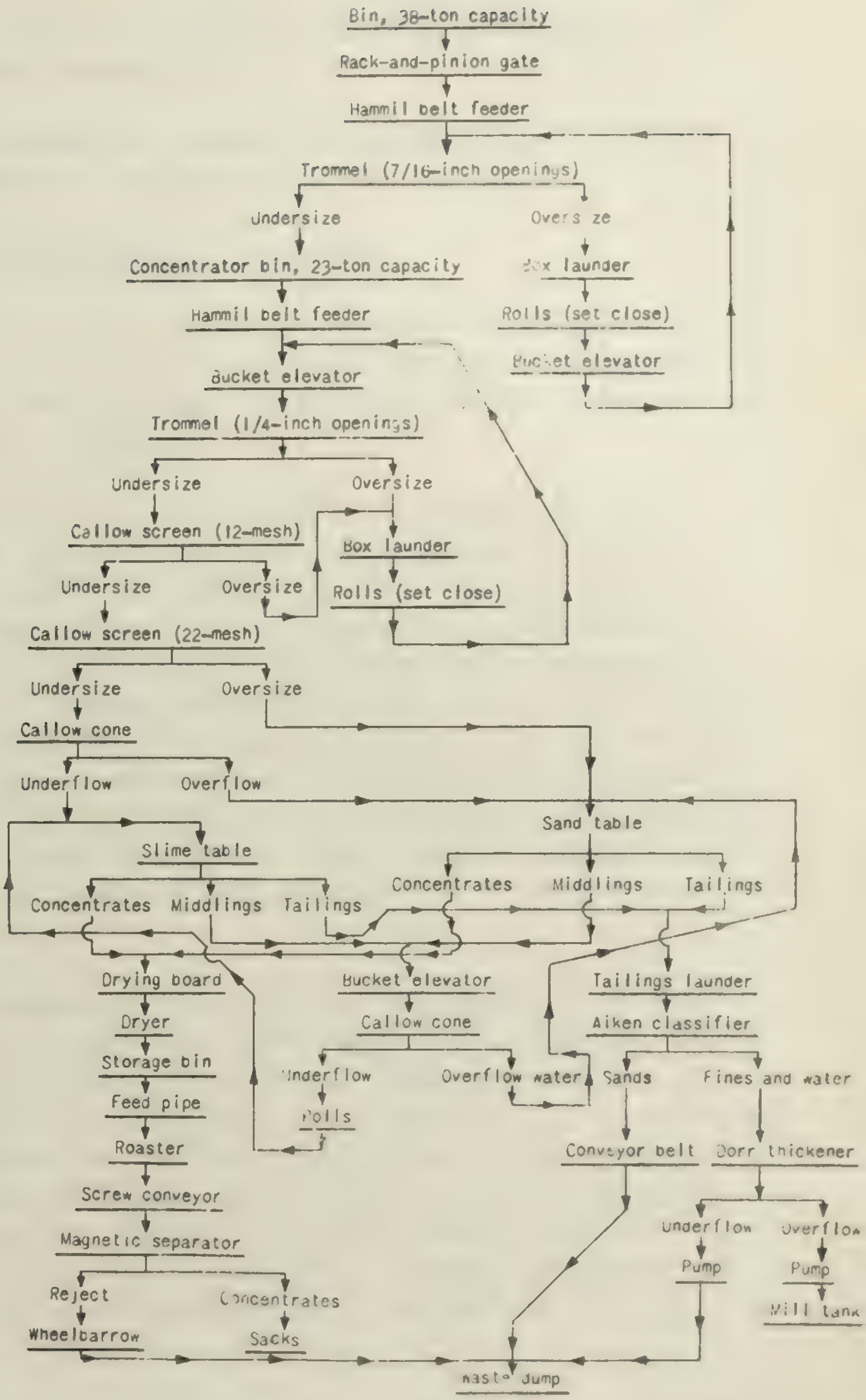

Figure 2.- Flow sheet of the concentrator

5. The table middlings are crushed further by rolls and returned to the table which treats the finer-sized material.

A flow sheet of coarse and intermediate crushing operations is given in Figure 1 and that of the concentrator is presented in Figure 2.

Coarse and Intermediate Crushing

The broken ore from the stopes is delivered to the mine tram bin in trains of eight 1-ton capacity cars hauled by a storage-battery locomotive; cars are dumped by hand. The mine tram bin has a capacity of 90 tons and is constructed of wood with the bottom sloping 45°.

The discharge of ore from the bin is controlled by a hand-operated rack and pinion gate. From the bin the material is fed to a grizzly 5 1/2 feet long and 2 feet wide, inclined 40°, and equipped with tapered 3/4 by 3/8 by 2 inch steel bars spaced 1 inch apart. The oversize is crushed to 1 inch size by a 16 by 24 inch Blake-type crusher equipped with corrugated manganese steel wearing plates. The crushed product joins the grizzly undersize and is delivered to a storage bin by a bucket elevator. The elevator is 35 feet long, center to center, and operates at a speed of 50 feet per minute and at an incline of 65°; it is provided with a 6-ply rubber-surfaced belt and buckets which are 8 inches long, 8 inches wide, and 6 inches deep. The storage bin is of wood and has a bottom which slopes 45° in two directions; it has a capacity of 200 tons.

The minus 1-inch product is transported from this storage bin to the mill by an aerial tramway 1,420 feet long; the difference in elevation between the upper and lower terminals is about 500 feet. The tramway is of the single-cable type and is equipped with 1-inch cable made up of 6 strands, 19 wires each, and a hemp core. There are 40 buckets in the line, each having a capacity of 240 pounds. The buckets are loaded while the tramway is in motion by a measuring hopper which operates on an inclined plane. The measuring hopper is filled from the bin by means of a hand-operated overcut-arc gate. The buckets are dumped automatically at the mill by a tripper which releases the hinged bottoms.

The 60-ton capacity receiving bin at the mill is of wooden construction and has a 45° inclined bottom. From this bin the ore is delivered through a rack and pinion gate to a grizzly by an eccentric shaker feeder. The feeder is 5 feet long by 1 foot deep and has a slope of 2 inches per foot; it is driven by chain at a speed of 40 oscillations per minute and with a 4-inch stroke. The grizzly is 4 feet long by 16 inches wide and is made of 3/4 by 3/8 by 3 inch tapered steel bars spaced 3/4 inch apart. The grizzly oversize is crushed to 3/4 inch by a secondary 14 by 18 inch Blake-type crusher equipped with corrugated, white, cast-iron wearing plates. The crusher is operated for approximately four hours per day when running the mill on two shifts. The crushed product joins the grizzly undersize and passes to a 38-ton capacity wooden storage bin of which the bottom slopes at 45°.

Final Crushing and Screening

Referring to Figure 2, the ore from the 38-ton capacity bin is fed to a trommel by a Hammil belt feeder; the trommel is made of 3/16-inch plate having 7/16-inch holes. The feeder belt is 12 inches wide and 18 inches long between pulley centers and travels at a speed of 90 feet per minute. The flow of ore to the feeder is controlled by a rack and pinion gate. The trommel is 4 feet long and 2 feet in diameter; it is inclined at 20° and operates at a speed of 21 r.p.m. in closed circuit with a pair of Allis-Chalmers 14 by 30 inch primary rolls. The trommel oversize is delivered to the rolls by gravity, and the crushed product is returned to the trommel by a bucket elevator. The speed of the rolls is 125 r.p.m. and that of the bucket elevator is 295 feet per minute. The elevator is 29.5 feet between pulley centers, vertically; it is equipped with a 10-inch 6-ply belt having a 1/16-inch rubber-top cover and with buckets 9 inches long, 6 inches wide, and 5 inches deep spaced on 18-inch centers.

The undersize from the 7/16-inch trommel passes by gravity to a 23-ton capacity wooden surge bin equipped with a 45° sloping bottom. From this bin a second Hammil belt feeder delivers the material to an elevator; the latter discharges into a second trommel equipped with 3/16-inch plate having 1/4 inch holes. The trommel is 4 feet long and 2 feet in diameter; it is inclined at 20° and operates at a speed of 19 r.p.m. and in closed circuit with a pair of 14 by 30 inch secondary Denver Engineering Works rolls. The oversize of the trommel is delivered to the rolls by gravity and the crushed product is returned to the screen by the wet elevator. The secondary rolls operate wet and at a speed of 125 r.p.m. The elevator is 28 feet long between pulley centers vertically and travels at a speed of 300 feet per minute; it is equipped with a 10-inch, 6-ply, 3/16-inch, rubber-surface belt and with cast-steel buckets 9 inches long, 6 inches wide, and 5 inches deep spaced 12 inches apart.

The undersize of the secondary trommel is delivered to a 4 by 2 foot simplex Callow screen equipped with 12-mesh phosphor-bronze cloth which travels at a speed of 70 feet per minute. The oversize is at present returned to the secondary rolls, although it is planned to install a third set of rolls to operate in closed circuit with this screen. The undersize is conveyed to a second 4 by 2 foot simplex Callow screen; the latter is equipped with 22-mesh phosphor-bronze screen cloth which travels at a speed of 60 feet per minute. The oversize and undersize products of this screen comprise the feeds to the sand and slime tables, respectively.

Concentrating

The plus 22-mesh sands and the thickened minus 22-mesh fines are each treated on one Universal Overstrom table; each table produces concentrates, middlings, and waste tailings. The middlings from both tables join and are dewatered in a 4-foot Callow cone; the cone discharge sands are further crushed by a pair of 18 by 4 inch Sturtevant middling rolls and the crushed product is returned to the table treating the undersize of the 22-mesh screen.

The table concentrates are collected in buckets which are emptied from time to time on a 4 by 7 foot drain board. The tailings are conveyed by launder to the dewatering plant.

The Callow cone which thickens the 22-mesh screen undersize before table treatment is 8-foot size and is equipped with a gooseneck discharge; the overflow water from the cone is used as feed water on the sand table.

The elevator of the circuit used for recrushing of table middlings is 18 feet long vertically from center to center of pulleys; it is equipped with an 8-inch belt having buckets 7 inches long, 5 inches wide, and 4 1/2 inches deep spaced 18 inches apart.

Cleaning of Table Concentrates

The table concentrates are shoveled by hand from the drain board into a Channon foundry sand dryer. The concentrates from one week of operation are dried in about three days using wood as fuel. The dried concentrates drop into a concrete storage bin which is 6 by 7 feet in section and 4 feet high.

From the storage bin the product is fed through a 2-inch pipe with plug valve control to a roasting furnace for the purpose of converting the contained pyrite to the magnetic sulphide. Referring to Figure 3, the furnace block, made of concrete, is 11 feet long by 4 1/2 feet wide; it is 6 feet 7 inches high at the discharge end and 8 feet 7 inches high at the feed end. A firebox 18 by 18 inches in section lined with fire brick is built into the upper part of the concrete block; the slope of the firebox is 2 inches per foot. Between the fire brick and the concrete is an insulating layer of diatomaceous earth 3 inches thick. The top of the firebox is covered with flat 15 by 24 by 3 inch tile. The tile are dapped into the brick.

The roaster tube consists of a 12-foot length of 8-inch standard iron pipe equipped with flanges at either end. These flanges rest on flanged rollers, the bearings of the latter being bolted to a steel plate resting on the roller blocks. The tube is driven by chain and gear at a speed of 11 r.p.m and is connected at the feed end by a hood to an 8-inch diameter stack for removal of the sulphurous gases. The joint between the roaster tube and the hood is sealed with fire clay.

The firebox is heated by an oil burner placed at the discharge end of the tube; plus 27° Baumé oil is atomized by 6-ounce air pressure and burned at the rate of 2 gallons per hour.

The concentrates as fed to the roaster contain about 19 per cent of pyrite; the latter is rendered magnetic in the presence of air, the time of roasting is about 5 minutes. The calcines discharge into a trough of a screw conveyor and are conveyed to a shaker-type feeder at the head of a Dings "M-M" magnetic separator. The conveyor is 16 feet long and is equipped with cast-iron flights and a sheet-steel trough. The primary purpose of the conveyor is to allow the concentrates to cool before the magnetic separation. The separator operates at

120 volts and with a current of 12 amperes. The magnetic material is rejected as waste; the cleaned concentrates are packed for shipment in double canvas and burlap sacks which hold about 120 pounds each.

DEWATERING AND DISPOSAL OF TAILINGS

The table tailings pulp is conveyed by a launder to an Aikens classifier which is 11 feet long and set with a slope of 2 1/2 inches per foot. The classifier sands are conveyed to the waste dump by a conveyor which is 99 feet long between centers; the belt is 16-inch, 8-ply canvas and rubber, with a 1/16-inch rubber-top cover. The classifier overflow pulp is fed to a 13 1/2 by 8 1/2 foot Dorr thickener for reclaiming of water. The thickener rakes travel at a speed of 1 revolution in 8 minutes. The thickener underflow is delivered to the waste dump by a 1-inch eccentric-driven diaphram pump; the thickener overflow is delivered to the 5,760-gallon capacity main supply tank by a 1 1/2-inch centrifugal pump.

As previously noted, the amount of ground available for the storing of tailings is limited because of the fact that the mill is located in a narrow canyon which has steeply sloping sides. The road to the camp follows the bottom of the canyon, and the expense of building another road at a higher elevation is prohibitive. Eventually, when additional water is made available by the installation of a pipe line from Spearmint Canyon the tailings will be sluiced to a point several miles below the mill site where the canyon is wider.

LAUNDERS

Due to the abrasive action of the mill pulp, the launders are lined on the bottoms with steel plate or with armorite; the latter is a high-quality rubber vulcanized to a fiber base. The bottom of the launder which conveys the mill tailings to the pump is protected with block riffles made of 2 by 4 inch material and placed 12 inches apart.

CONTROL OF OPERATIONS

A rapid and reliable method for the determination of WO_3 in tungsten ores by chemical methods has not been devised. With the present standard methods in use it is difficult to obtain accurate results unless extreme care is exercised in manipulation. Because of the expense involved and the time required for accurate chemical determinations, mill operations at Silver Dike are controlled by panning. This method has proved rapid and reliable at Silver Dike when done by an experienced operator. A 6-inch frying pan is used for the estimation.

A specific gravity method for the approximate determination of tungsten trioxide content has been used in the Boulder and Atolia tungsten districts, but such a method is not applicable when the ore contains heavy minerals other than scheelite or when the specific gravity of the gangue minerals varies.

MARKETING OF CONCENTRATES

Concentrates produced by the Nevada-Massachusetts Co. are sold with a quality guarantee similar to the tabulation which follows:

Tungsten trioxide ...	per cent ...	65 to 70
Tin	per cent maximum	Trace
Copper	do	0.05
Arsenic	do	.035
Sulphur	do	.75
Antimony	do	.035
Phosphorous	do	.05
Bismuth	do	.035

The only undesirable elements contained in the shipping concentrates from Silver Dike are sulphur and copper, and the amounts of these elements are well within the limits prescribed by the quality guarantee under which sales are made.

The Nevada-Massachusetts Co., Inc., maintains its own selling agency. Samples of carload lots of concentrates are sent to custom assayers by buyer and seller and are sold on the analyses thus reported if within the limits previously agreed upon. If not within the prescribed limits, another sample is sent to an umpire assayer, and the sample which has an analysis closest to the result reported by the umpire is averaged with the umpire assay for the basis of sale.

EXPERIMENTAL WORK WITH FLOTATION METHODS

Preliminary experimental work by the Bureau of Mines at Rolla, Mo., on Silver Dike ore by flotation methods indicates that scheelite is floated readily. A sample weighing 200 pounds was crushed to minus 10-mesh in an Abbe mill. The pulp was then treated in a mechanical-agitation type of flotation machine which produced rough concentrates and tailings. The concentrate were cleaned twice, the latter operations producing finished concentrates and middlings. The tabulations which follow give the results of this test and the quantities and the kinds of reagents used.

Results of experimental flotation test

	Weight per cent	Assays per cent WO_3	Per cent of total WO_3
Flotation concentrates ..	3.4	62.72	89.7
Flotation middlings	21.7	1.03	9.4
Flotation tailings	74.9	0.03	0.9
Composite	100.0	2.38	100.0

Flotation reagents used

	Weight per ton of ore treated, pounds		
	Roughing	Cleaning	Total
Pine oil	0.12	---	0.12
Oleic acid	.16	---	.16
Sodium oleate	.40	---	.40
Sodium carbonate	.75	0.75	1.50
Sodium silicate	--	.40	.40

It has been found that the amount of reagents can be varied widely without seriously affecting either the grade of concentrates produced or the recovery of WO_3. In one test the concentrates produced contained 79 per cent of WO_3 as compared to 62.72 per cent in the preceding tabulation; the recovery of WO_3 in the former test was but slightly less than that obtained in the latter.

An analysis of milling losses at Silver Dike indicates that the chief loss is not due to included grains but is due to the inability of tables to make a high recovery of the slimed scheelite. The method of crushing used at Silver Dike employs two stages of Blake-type crushers followed by three stages of rolls; this practice is designed to prevent the sliming of scheelite and thereby decrease the loss in the tailings of the table treating the finer portion of the ore. The experimental flotation work described indicates that the treatment of Silver Dike ore by flotation methods is feasible and that the successful application of these methods would not only simplify the preparation of the ore for concentration by eliminating a number of the crushing stages but would also increase the recovery of the WO_3.

LABOR

When the mill is operated on a 2-shift basis, five men are required to run the mine and mill crushing plants, tables, roaster, aerial tramway, and accessory equipment. Two men are employed on the rolls and crusher at the mill, two are required for the tables, magnetic separator, and roaster, one man on the day shift operates the mine crusher, aerial tramway, and does general repair work about the mill.

The wage scale for mill labor is $5 per shift of eight hours.

METALLURGICAL DATA

Metallurgical data for a two months period in 1931 are given in Table 1.

Table 1. - Metallurgical data

Ore milled	.... dry tons	1,140
Moisture content of ore	.... per cent	3
Days operated	....	48
Hours operated per day	....	16
Average ore milled per day	.... dry tons	24.2
Average WO_3 content in mill feed	.... per cent	0.8
Average WO_3 content per ton of mill feed ...	.... pounds	16
Concentrates produced	.... do	20,062
Analysis of concentrates:		
Tungsten trioxide	.... per cent	73.38
Sulphur	.... do	0.17
Copper	.... do	0.003
WO_3 produced in concentrates	units of 20 pounds	736.1
WO_3 lost in tailings	do	126
WO_3 recovered (estimated)	.... per cent	80.7
Ratio of concentration, tons into 1	....	114
Net water consumption per ton of ore treated	.... tons	1.2

COSTS

A summary of milling costs for a two months' period in 1931 is given in Table 2; distribution of labor, power, and supplies is indicated in Table 3.

Table 2. - Summary of milling costs

	Cost per ton of ore treated 1/			
	Labor	Power	Supplies	Totals
Aerial tramway	$0.104	$0.135	--	$0.239
Crushing to 3/4 inch	.104	.271	--	.375
Crushing, screening, and conveying	.339	.422	$0.276	1.037
Tabling	.208	.082	--	.290
Roasting, magnetic separation, sacking	.104	.033	.162	.299
Dewatering and pumping	.130	.135	--	.265
Supervision	.066	--	--	.066
Miscellaneous	.052	--	.026	.078
Totals	1.107	1.078	0.464	2.649

1/ Direct costs only.

Table 3. - Distribution of labor, power, and supplies

Labor (per ton of dry ore):		
Aerial tramway	man-hours	0.151
Crushing to 3/4 inch	do	.168
Crushing, screening, and conveying	do	.540
Tabling	do	.334
Roasting, magnetic separation, sacking ..	do	.184
Dewatering and pumping	do	.209
Miscellaneous	do	.084
Total	do	1.670
Ore treated per 8-hour man-shift	tons	4.78
Power (per ton of dry ore):		
Aerial tramway	kilowatt-hours	4.45
Crushing to 3/4 inch	do	8.91
Crushing, screening and conveying	do	13.83
Tabling	do	2.66
Roasting, magnetic separation	do	1.08
Dewatering and pumping	do	4.44
Total	do	35.37
Labor, part of total cost	per cent	41.9
Power, part of total cost	do	40.7
Supplies, part of total cost	do	17.4

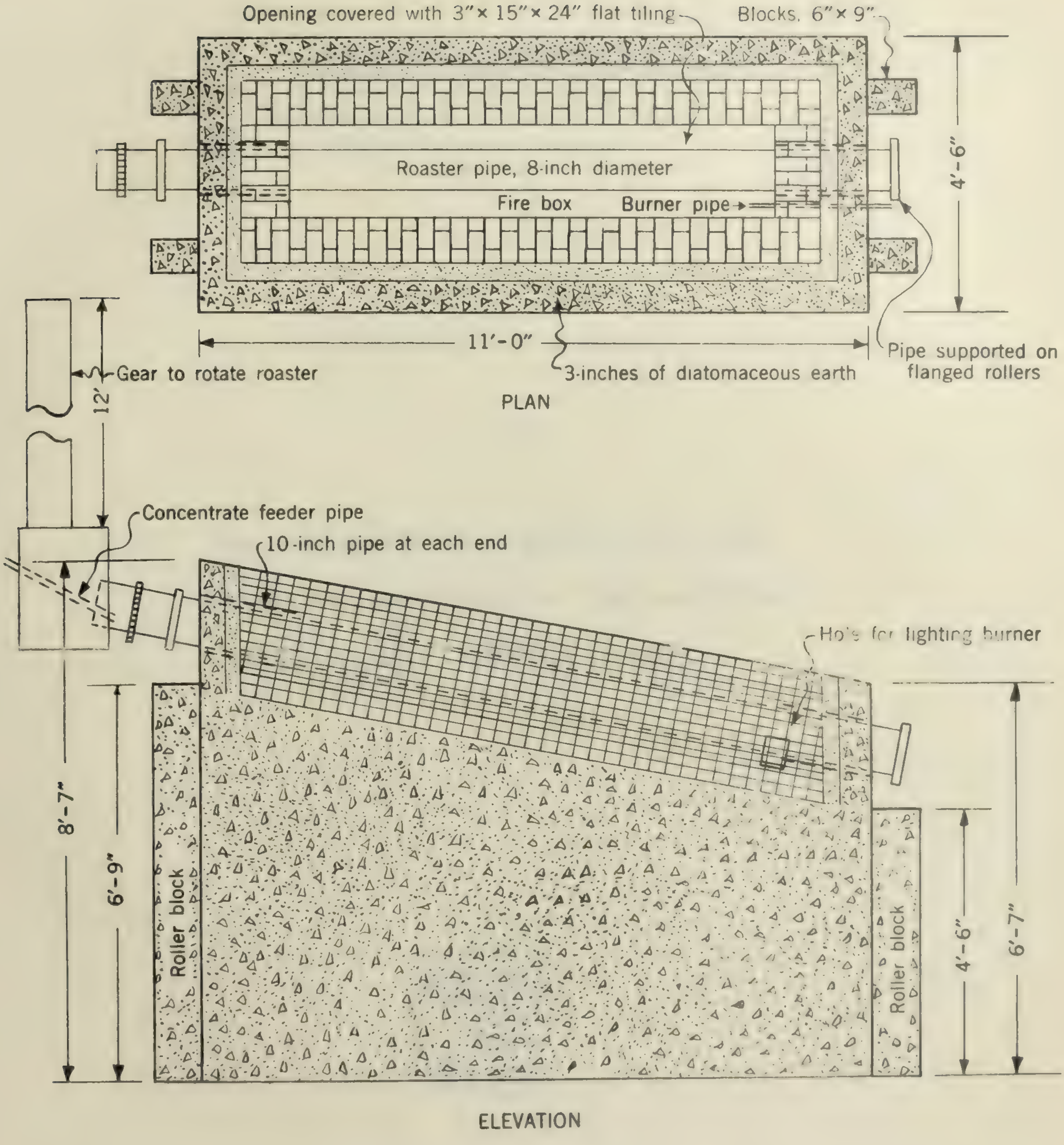

Figure 3.—Plan and elevation of roaster

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MILLING METHODS AND COSTS AT THE PECOS CONCENTRATOR OF THE AMERICAN METAL CO., TERERRO, N. MEX.

BY

H. D. BEMIS

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE – BUREAU OF MINES

MILLING METHODS AND COSTS AT THE PECOS CONCENTRATOR OF THE AMERICAN METAL CO., TERERRO, N. MEX.[1]

By H. D. Bemis[2]

INTRODUCTION

This paper describing the milling practice of the American Metal Co. at its Pecos concentrator is one of a series on milling methods and costs being prepared by the United States Bureau of Mines.

ACKNOWLEDGMENT

The author acknowledges the assistance rendered by Guy Martin, metallurgist, H. E. Brown and J. H. Hastings, chemists, and Thomas J. Fahey, accountant, in gathering and preparing the data presented in this paper.

GENERAL

The Pecos mill treats complex silver and gold bearing lead-zinc-copper ore by selective flotation methods at the maximum rate of 600 tons per day and produces lead concentrates and zinc concentrates.

It is located near the western boundary of San Miguel County, N. Mex., about 2 miles north of U. S. Highway 85. Freight is received and delivered over a 4-mile spur which connects with the main line of the Atchison, Topeka, and Santa Fe Railroad at Glorieta.

The mine is 12 miles north of the mill and is connected with it by an aerial tramway.

Water for milling, power house, and domestic uses is pumped from the Pecos River through a steel pipe line 8 inches in diameter and 7,000 feet long; the head amounts to 600 feet. The water is chlorinated for domestic use by the addition of calcium hypochlorite to the pump sump of the pumping plant at regular intervals.

Power is supplied to the mill on a 3-phase, 60-cycle, 2,200-volt circuit from the company power house located 600 feet distant from the mill. For use in lighting and for small motors it is stepped down to 110 and 440 volts, respectively.

ORE TREATED

The ore is mined by the square set and fill method. It is, in general, massive sulphide material which grades to a mixture of sulphides and gangue materials. The ore bodies are replacements in pre-Cambrian schists and associated crystalline rocks. The sulphides, in order of abundance, are pyrite, sphalerite, galena, chalcopyrite, marmatite, chalcocite, bornite, argentite, and proustite; the gangue minerals comprise chiefly mica, quartz, tourmaline, and hornblende.

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1 The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from U S Bureau of Mines Information Circular 6605 "

2 Mill superintendent, Pecos concentrator and consulting engineer, U S Bureau of Mines

The sulphide grains range from 1/8 inch in diameter to extremely fine particles which require magnifications of from 300 to 1,200 diameters for identification under the microscope. The nature of the occurrence of gold in the ore is not definitely known, but it is certain that it does not exist as free gold. Determinations have indicated a considerable segregation of the gold with the coarse mica. Screen analyses of very clean mica show that although the average gold content in this material is quite small, a considerable concentration of it occurs in the plus 100-mesh size; the plus 48-mesh size contains a surprisingly high gold content varying from 0.06 to 0.15 ounce per ton.

The grinding index figure of this ore in the Lennox series is from 83 to 84; this is based on Cripple Creek phonolite taken as 100.

A typical analysis of the mill feed shows metal values as follows:

Gold............ounces per ton	0.10
Silver......... do.	3.3
Lead........................per cent	4.9
Copper.................... do.	0.8
Zinc........................ do.	15.4

GENERAL ARRANGEMENT OF PLANT

The concentrator was designed to treat 600 tons of ore per 24 hours; it was built in 1926 at a cost of $381,000. Changes and additions since that time have increased the total cost to $410,000. The plant is built on a hillside, which allows the main pulp stream to flow by gravity. The floor space is arranged in eight bays, which accomodate the following groups of equipment: (1) Grinding and classifying; (2) mica flotation, mica cleaning, and mixing of chemicals; (3) lead-circuit conditioners, lead-flotation roughers and cleaners, and No. 1 zinc-circuit intermediate conditioner; (4) zinc-flotation roughers and cleaners and No. 2 zinc-circuit intermediate thickener; (5) zinc-flotation scavengers and tables for zinc treatment; (6) pilot tables and sample driers; (7) lead and zinc concentrate thickeners; (8) filters and concentrate bins.

The front of the mill ore bin forms the back of the grinding bay and the mill terminal of the tramway is carried on the top of this bin. The filters are mounted on top of the concentrate bins. Concentrates are shoveled by hand from the bins into chutes which convey them to a loading machine in the box car situated below and in front of the bins. The vacuum pumps and blowers are placed on an extension of the lead circuit flotation floor at one side of the main mill building. Chemicals and general supplies are stored in a warehouse which is connected with all floors of the concentrator building by an inclined tramway.

Two 125,000-gallon capacity mill water tanks are located well back of the mill, on the hillside, at an elevation of about 20 feet above the top of the tram housing. About 50 feet above the mill water-supply tanks are two 50,000-gallon capacity fire-protection water tanks. All water received at the mill site must pass through the latter, the overflow going by gravity to the larger main tanks.

The fire tanks are connected exclusively to a Rockwood sprinkler system in the mill and to the outside hydrants surrounding the mill, warehouse, and offices. In addition to the sprinkler system the mill is protected with inside hose lines connected to the mill water-service lines, and pyrene extinguishers are located on all floors and in adjacent buildings. All tanks are fitted with mud traps and washout lines.

HISTORY OF CONCENTRATOR OPERATIONS

The mill started to operate on January 1, 1927, using a flow sheet which was evolved from considerable test work done at the San Francisco laboratory of the Minerals Separation Co. Large continuous-test runs were made and final decisions concerning the flow sheet were based on the results of these tests. Several tons of ore out of the San Francisco shipment were then sent to the Colorado Experimental Plant at Golden, Colo., and the principle tests made at San Francisco were repeated, using different operators and somewhat different conditions. The results of test work conducted at the two plants checked quite closely and no changes in the general treatment resulted from the work done at Golden. It was, however, decided from the results of the testing at Golden to increase the time of pulp conditioning to some extent.

The original plans called for the depression of the mica, which is present in considerable quantity in most of the ore. This was finally abandoned in favor of its removal by selective-flotation methods.

The next important change involved the use of zinc sulphate in conjunction with cyanide as a zinc depressant. Original tests did not indicate the need of this reagent, and the mill had operated over a year before it was adopted, early in 1928. In 1929 finer grinding was adopted, although all of the ore does not require it. At the beginning of 1930 soda ash and ethyl xanthate were replaced by lime hydrate and amyl xanthate in the lead circuit.

Since the beginning of milling operations in 1927 a gradual transition has taken place in the length of time given to flotation treatment. The time has been lengthened by allowing longer periods for the conditioning of pulp and by the addition of scavenger flotation cells in the zinc circuit. Improved recoveries and the use of smaller amounts of reagents have resulted from this change.

Cleaner flotation cells were added to the zinc circuit in 1930, and although an economic advantage resulted in the production of higher-grade zinc concentrates, the recovery of zinc decreased. The tabulation which follows indicates the changes that have taken place, since operations started, in the grades of concentrates produced and in lead and zinc recoveries.

Year	Ore treated, tons	Lead concentrates		Zinc concentrates		Milling cost per ton of ore treated
		Lead, per cent	Lead recovery, per cent	Zinc, per cent	Zinc recovery, per cent	
1927	162,175	28.33	77.44	50.67	87.18	$1.499
1928	201,013	34.81	76.86	50.77	85.50	1.440
1929	216,809	36.62	79.54	52.22	86.68	1.212
1930	151,943	37.81	81.97	54.45	84.64	1.193

PRESENT METHOD OF CONCENTRATING

Coarse Crushing

The ore is crushed to minus 1½-inch size in the coarse-crushing plant located at the mine. The product as received at the concentrator contains about 2 per cent of moisture; a typical screen analysis of this material follows.

Screen analysis of ore as received at the concentrator

	Weight, per cent
Plus 1½-inch	0.6
Plus 1-inch	5.4
Plus ½-inch	34.9
Plus 3/16-inch	21.1
Plus 10-mesh	10.5
Plus 20-mesh	4.9
Plus 35-mesh	5.8
Plus 65-mesh	4.6
Plus 150-mesh	3.7
Plus 200-mesh	1.1
Minus 200-mesh	7.3
Total	99.9

Concentrator

The concentrator contains three grinding units, two in operation and one spare, and two flotation units with the necessary accessory dewatering and sampling equipment. The flow sheet of ore treatment is presented in Figure 1.

Grinding

Ore from the coarse-crushing plant, as previously mentioned, is delivered to the concentrator ore bin by an aerial tramway. This bin is constructed of wood and has a flat bottom; it has a total capacity of 1,800 tons and a live capacity of 1,200 tons. Six steel hoppers, each equipped with an 18-inch rank and pinion gate, are placed along the bottom of the bin and direct the ore onto six feeders, two of the latter being provided for each grinding unit. The feeders are 24-inch heavy-duty Webster pan conveyors with 6-inch sides and operate at a speed of 29 inches per minute; the speed, however, is subject to variation by means of an adjustment lever. The power input to each feeder amounts to 1 1/3 hp. The only repairs on the pans and rollers during four years of service have been in connection with a few loose side plates, a few broken springs, the occasional change of a lever pin bushing, and a few sets of pan return rails.

A description of one of the three grinding units follows. The two feeders deliver the material to one No. 75 Marcy ball mill which operates in closed circuit with one duplex Dorr classifier, 6 feet by 23 feet 8 inches in size.

The ball mills are driven at a speed of 22.5 r.p.m. by 150-hp. General Electric supersynchronous motors. When fed at the rate of 256 tons per day the power input to the motor is 151 hp.

Manganese steel liners and chrome molybdenum steel liners have been used. The life of one set of the former is 3,939 operating hours, which is equivalent to 42,000 tons of ore ground. Experimental work on the latter material covering various steels and different heat treatments has been in progress about four years; the results of this work indicate an average life of 4,304 hours for this material which is equivalent to 46,000 tons of ore ground. The average consumption of liners amounts to about 0.3 pound per ton of material ground, including the weight of scrap liners discarded.

Forged steel balls 4½ inches in diameter are used; the ball consumption averages 1.8 pounds or $0.07 per ton of material ground.

The cost of liners, scoops, and miscellaneous ball-mill repairs averages $0.07 per ton of ore handled.

The Model D Dorr classifiers are set with a slope of 3¼ inches per foot; each classifier is driven with a rake speed of 22 strokes per minute by belt from a line shaft which in turn is connected through a Texrope drive with a 10-hp. motor. This motor also drives the two feeders of the unit; the power input to the motor is 4.4 hp., the classifier accounting for 3.73 hp. Circulating loads of 400 per cent are carried by the classifiers. Rakes have an average life of about two years; those of the upper sections last about three years and those of the lower sections about one and a half years.

The classifiers are equipped with chip rakes and screens for intercepting the wooden chips that are always present, owing to the large amount of mine timber used. Approximately 50 per cent of the wood is eliminated on the picking belt in the coarse-crushing plant. The remainder is pulverized in the grinding mills and becomes an intolerable nuisance throughout the mill unless removed. The classifier chip screens have removed 2,536 cubic yards of this material in four years, during which time the ore milled amounted to 731,940 tons; this amounts to 1 cubic foot of chips removed for every 10.7 tons of ore treated. Put in another way, each classifier screen removes about 1 cubic foot of tightly packed wooden pulp and chips per hour.

The tabulation which follows gives screen analyses and pulp densities of ball-mill discharge, sand, and overflow products.

Screen analyses and pulp densities of
ball-mill discharge, sand, and overflow products

	Solids, per cent	Weight, per cent					
		Plus 48	Plus 65	Plus 100	Plus 150	Plus 200	Minus 200
Ball mill discharge	85.0	17.2	15.3	13.2	15.7	11.8	26.8
Classifier sands	83.0	20.0	18.1	13.7	18.5	13.5	16.2
Classifier overflow	32.0	-	2.2	3.6	5.6	7.8	80.8

In the classifier overflow all of the plus 65-mesh and most of the plus 100-mesh material is mica gangue, and, due to the variable amounts of this mineral present in the ore, the grinding is controlled by the amount of minus 200-mesh material produced, rather than by the amount remaining on a coarser screen.

Distributing Feed to the Flotation Units

The classifier overflow pulps of the grinding units join in a common launder for conveyance to the pulp distributor. The latter is a rotating cast-iron tub, 24 inches in diameter and 18 inches deep; it is equipped with four 4-inch holes placed practically at the bottom and spaced equidistant around the side. The holes are threaded, bushed down to 2½ inches, and equipped with nipples and 90° ells; the ells face backwards from the direction of rotation and downwards at about 45°. This tub is fed by a vertical tube built around the supporting shaft and into which is fitted the launder which brings the combined classifier overflow pulps. This tube extends to within 6 inches of the bottom of the tub and usually maintains a pulp depth of 12 inches in the tub. It has been found that the tub will not give equal distributions of the pulp through the ports if fed by a launder which discharges

into one side of it, as with this arrangement the ports in front of the pulp stream discharge more material than those which are passing under the end of the launder. Satisfactory distribution is obtained when uniform pulp is available at equal pressures at all discharge ports. Furthermore, if the discharge ports are too large, the pulp will take the easiest path out of the distributor, and under these conditions there is no certainty that all ports are discharging equal or practically equal volumes of material.

The distributor tub revolves at a speed of 18 r.p.m. and is hung by its shaft in a round wooden catch tub; the bottom of the latter is divided radially into four compartments by 2 by 6 inch timbers placed at right angles to each other at the center of the tub; each compartment is equipped with a 5-inch pipe flange and nipple discharge placed at the bottom. The pulp from the revolving tub is discharged at a minimum velocity and drops into the four receiving compartments with little splashing. The pulps from two opposite compartments join together as feed to one of two flotation units.

Flotation of Micaceous Material

Each of the two mica flotation units is provided with one 10-cell 18-inch Minerals Separation Sub A machine which produces rougher mica concentrates and tailings. The mica concentrates from both rougher units join and after being diluted with water are further treated in one air-lift type of cleaner cell. The latter produces mica concentrates which are treated further by tables, and tailings which join the primary feed of the mica rougher cells.

Four Wilfley tables are used for the cleaning of mica concentrates; these are arranged in pairs, two in each mill unit, and the concentrates from the cleaner cell are divided equally between the pairs in order to balance the metallurgical accounts of the two general mill flotation circuits. Each pair of tables is arranged in series, the second table retreating the tailing of the first; the concentrates of both tables comprise sulphide minerals brought over with the mica froth in the flotation cells and are added to the lead concentrates produced in the lead flotation units. The table concentrates would be added to the head of the lead flotation circuit if enough were produced to justify the installation and operation of two pumps which would be required for this work.

The table tailings contain the bulk of the mica and are rejected as waste. As discarded, the mica material contains metal values well below the general tailings, except for the zinc content, which is about equal to that contained in the usual tailings. The tabulation which follows gives typical metal contents of the discarded material.

Typical metal contents of discarded material

Gold..............ounces per ton	0.01
Silver..............do.....per ton	0.34
Lead...........................per cent	0.56
Copper.................. do.	0.08
Zinc...................... do.	1.50

The only reagent used in the mica flotation circuits is pale cresylic acid and this is added to the rougher cells at the rate of 0.13 to 0.15 pound per ton of ore treated.

The motors used to drive the impellers of the mica roughers are standard horizontal motors and although placed vertically have not developed bearing troubles in four years of

operation. The ball bearings at the top of the flotation machine spindles require changing at about 2-year periods. This bearing trouble seems to be caused mainly, if not entirely, by the hard residue of the grease used, perhaps a wax residue. It is hoped that the use of one of the later forms of low wax greases will greatly extend the life of these bearings.

The impellers of the mica roughers are driven at a speed of 325 r.p.m. and when treating a pulp containing 32 per cent of solids, which amounts to a total of 256 tons of solids per day, each impeller motor requires 2.87 hp.

Lead-Flotation Circuit

The tailings pulp of each mica-flotation circuit is conditioned in an 8 by 12 foot Deveraux agitator. Lime and zinc sulphate are added to the agitators at the rates of 0.6 and 0.78 pound per ton of ore treated, respectively; the time of conditioning is about 15 minutes. The conditioned pulp from each agitator is treated in one 16-cell, 18-inch Minerals Separation Sub A machine; the first four cells are operated as cleaners and the remaining twelve as roughers. The feed pulp enters the fifth cell of the machine; rougher lead concentrates are produced from the first nine rougher cells, and these, with an additional 0.4 pound of zinc sulphate per ton of mill heads, are pumped to the first cleaner cell. The middling froths of the last three rougher cells are returned to the ninth rougher cell. Some of the cleaner cells produce finished concentrates and the remainder produce either finished concentrates or middlings, depending upon the grade of the heads. Middlings, when produced, are returned to the head of the cleaner unit. Two launders are provided in front of the lead cleaners; the inner one leads to the rougher froth pump; the outer one receives finished lead concentrates and is extended to receive concentrates from the first two rougher cells. This arrangement provides a means of discharging concentrates when the feed pulp unexpectedly increases in lead and copper contents; on the other hand, a run of unusually low-grade ore is handled by allowing the last one or two of the four cleaner cells to discharge froth into the inside launder by removing the pans which usually convey this froth to the outer launder.

A 2-inch direct-connected Wilfley pump elevates the rougher froth from a pump pit to the head of the cleaner. The pulp as it enters the first cleaner cell contains variable amounts of solids but averages around 20 per cent.

Amyl xanthate is added to the conditioned pulp at the rate of 0.4 pound and sodium cyanide is usually added to the fourth rougher cell at the rate of 0.06 pound per ton of ore treated. No additional frothing reagent is used in the lead circuit, as the pulp contains enough residual cresylic acid from the mica circuit to produce the desired volume of froth. The finished lead concentrates go to the lead concentrate dewatering unit and the tailings comprise the feed to the zinc flotation unit.

The conditioning tanks at the head of the lead flotation units are necessary because neither lime nor zinc sulphate may be added to the pulps of the mica-flotation circuits; the zinc sulphate depresses the mica slightly and the lime depresses it seriously. The flotation of lead can not, however, be carried out successfully in the presence of sufficient lime to depress the mica completely. The use of the conditioners decreases the amounts of lime and zinc sulphate which it would otherwise be necessary to add by one-half and one-third, respectively; they also make the operation of the rougher units more uniform by smoothing out the variations of feed that sometimes come from the grinding units.

At this plant it has been found best to add all the xanthate required for the flotation of lead at the head of the rougher cells. Originally, this reagent was added by means of a 3-way feeder which distributed 50 per cent of it to the head of the roughers, and 25 per cent

each to the fifth and ninth rougher cells. With proper conditioning of the pulp before starting flotation operations there appears to be no advantage gained by the progressive additions of the activating reagent, and the latter method offers a disadvantage in that more feeders require attention. Addition of the xanthate to the conditioners has been tried, but no advantage resulted; the only disadvantage in this procedure is due to the fact that the amount of xanthate added is subject at times to considerable change and the time element involved in the conditioners is a hindrance in quickly judging the results of a change.

As already noted, the cyanide reagent is usually added to the pulp when the roughing operation is about half completed; at this point both iron and zinc minerals began to appear in the froth in appreciable quantities. Operating experience at this plant indicates that the addition of cyanide to the conditioners requires more cyanide and decreases the recoveries of gold and copper in the lead concentrates. Normally there is no pronounced difference in metallurgical results if the cyanide is added at the head or at the middle of the roughing operations; however, if the zinc and iron minerals begin to float rapidly in the lead circuit, the point of adding the cyanide is changed from the middle to the head of the rougher circuit; the usual condition is such that the cyanide is added as near the end of the rougher as possible, and the point of addition is determined, as already noted, by the indications of zinc and iron minerals occupying a prominent position in froth contents.

Nearly all improvements made in the metallurgical results of treating this ore have been based on increased recoveries of gold and copper in the lead concentrates, and these improvements have been accomplished by reducing the amount of cyanide used; it is possible that further progress will be made along this line.

Under normal operating conditions the ore is received at the mill soon after being broken. When mine production decreases, however, crushed ore is drawn from the backs of the tram bin at the mine and the mill bin is also drawn down. The treatment of this material gives some trouble due to increased activity of the pyrite. If the amounts of lime and cyanide added are increased to overcome the increased activity of the pyrite, there is a slight decrease in lead recovery and also decreased recoveries of gold and copper in the lead concentrates. The metallurgical results of the zinc circuit, however, are seldom, if ever, affected.

The Deveraux conditioners are operated with about a 10½-foot depth of pulp; the feed enters at the top. They are equipped with 30-inch diameter impellers which are driven at a speed of 148 r.p.m. by the same gear heads as are used on the gear-driven 18-inch Minerals Separation machine. The two 8 by 12 foot machines are driven by a 10-hp. motor through a line shaft and belts; the motor requires an input of 6.75 hp. In order to prevent the agitators from trapping coarse material, they are discharged by means of a vertical riser, 8 by 8 inches inside dimensions, which is bolted to the inside of the agitator tank; the pulp enters at the lower end of the box at a point 3 feet above the bottom of the agitator, which is about at the level of the impeller and discharges through a hole placed in the side of the tank 11 feet above the bottom. Very little froth accumulates in the agitator, as the impellers are operated at sufficient speed to form a vortex into which the froth is drawn. This method of operating is quite necessary, as the formation of a froth bed in the agitator would result in uncertain treatment of the contained minerals.

The two Minerals Separation machines of the lead circuit are of the gear and line-shaft type of drive. Each is driven by a 50-hp. 2,200-volt motor and each motor requires a power input of 40.5 hp. when treating ore at the rate of 226 tons per day. One machine is driven from the motor through a Texrope drive and the other through a Link-Belt silent-chain drive.

An additional 6-cell Minerals Separation machine is provided for the lead circuit and is so placed that it may be used in series with either 16-cell machine. Its use improves

metallurgical results when feeds in excess of 250 tons per day per 16-cell machine are used. With present practice of feeding 250 tons of ore per machine per day and with present prices for metals any improved metallurgical results gained by the operation of this 6-cell machine are largely absorbed by extra operating expense. When feeding the 16-cell machines at the rate of 275 tons per day, the operation of the 6-cell machine is, however, of positive value.

Zinc-Flotation Circuit

The lead circuit tailings are conditioned for the flotation of zinc in two 12 by 12 foot Deveraux agitators equipped with 36-inch impellers which operate at a speed of 109 r.p.m. The two agitators, one serving each unit of the zinc circuit, are driven by belt from a line shaft which in turn is driven by a motor that requires a power input of 10.5 hp. The pulp, a few seconds before entering the conditioners, receives 0.7 pound of copper sulphate per ton of mill feed, and lime hydrate is added to the conditioners at the rate of 2 to 2.5 pounds per ton of mill feed. The older practice of adding both copper sulphate and lime to the conditioners required 1.0 pound of copper sulphate per ton of mill feed.

Thorough tests over considerable time periods failed to indicate any change in metallurgical results by the presence or absence of external air in either the zinc or lead conditioners. At the conclusion of these tests the air pipes of one zinc conditioner were connected to the steam lines and the temperature of the pulp gradually raised from the normal of around 60° to 180°F. No improvement was noted in operating results, other than a small reduction in the amount of frothing reagent required, and at 120°F. the pyrite began to give trouble, increasing with increase of temperature. The normal temperature of the pulp at the end of the zinc circuit roughers under present operating conditions is around 75°F. The increase in temperature of pulp as compared to former operations is due, first, to the smaller amounts of cold river water used at all points in the mill, and the substitution for it of power-house condenser water which enters the circuit at temperatures between 80 and 90°F. at the ball mills and classifiers; and, second, to the connecting of blower inlets with inside draft tubes which draw air from points near the mill roof. The objects of making the changes noted were to maintain the pulp temperature around 70°F. or higher during flotation operations and to reduce or eliminate the day and night and the seasonal fluctuations in temperature.

The conditioned pulp of each agitator passes to a 16-cell, 18-inch Minerals Separation Sub A machine. As operated at present the first five cells produce rougher concentrates which are cleaned in one 3-cell, 18-inch Minerals Separation unit. The remaining 11 cells produce froth products which are fed to a 20 by 12 foot Dorr thickener by a 3-inch Wilfley sand pump; the thickened pulp containing about 35 per cent of solids is returned to the zinc-circuit conditioner.

The cleaner froth passes over a Wilfley table enroute to the zinc thickeners. A high-grade gold-copper product is removed by the table and added to the lead concentrates.

The tailings of the cleaner are passed over a standard Wilfley table. The table produces concentrates, which join the finished lead-flotation concentrates due to high gold content, and tailings, which after thickening are returned to the conditioner at the head of the zinc circuit. This table functions in general as a pilot machine for the flotation cleaner unit.

The motors and drives of the zinc roughers are the same as described for the lead roughers. The 6-cell cleaner machine is divided into two 3-cell units, one serving each zinc rougher; each is driven by an individual Texrope drive. The power input per rougher cell is 2.5 hp., which amounts to 40 hp. per machine; the power input for the 6-cell cleaner tested as a unit is 1.63 hp. per cell.

Sodium ethyl xanthate and Yarmor pine oil are added to the feed launders of the zinc roughers at the rates of 0.07 and 0.08 pound per ton of mill feed, respectively.

The first six cells of each zinc rougher machine have enlarged spitz compartments about 8 inches wider than those of the standard cell; these are believed to be of advantage when large volumes of froth due to excess oil or air are to be handled in that they give additional time for the draining and the consolidation of the froth. The spitz compartments of the cleaner cells are not enlarged, as it is believed that no advantage is gained when the froth handled is dense and of reasonable volume.

The tailings of the zinc rougher are treated further in a 10-foot air-lift type scavenger cell with 0.09 pound of sodium ethyl xanthate per ton of mill feed. The original scavenger machine started as an experimental Forrester cell and was later changed to a Hunt air-lift machine, the original tank being utilized for this purpose. The tank was finally materially widened and equipped with froth paddles and in this form operates satisfactorily as a scavenger.

The scavenger concentrates are passed over a pilot table mainly for observation. The upper portion of the table streak consisting of clean pyrite is run to the tailing launder and the remainder of the table product is returned to the conditioner at the head of the zinc circuit.

The scavenger flotation cell produces very little material of commercial value, the concentrates consisting mainly of pyrite with small amounts of zinc minerals; the essential value of the machine is as a pilot. Much difficulty has been experienced in the use of Wilfley tables as pilots on zinc-circuit tailings due to streaks of dark gangue minerals resembling zinc minerals; this difficulty is especially troublesome at night. For this reason the scavenger flotation machine was substituted as a pilot and the froth treated on tables, since the troublesome gangue minerals will not float in the scavenger cell.

The scavenger machine tailings are conveyed by launder to a 7-compartment revolving distributor which formerly fed seven Wilfley tables. A narrow cut on the upper edge of the pyrite streak of each table was removed formerly and added to the flotation lead concentrates or its gold content. The gold contained in Pecos ore is not free, as previously stated, but occurs associated with some unknown mineral which is difficult to float and is concentrated to some extent by tables; when concentrated it is found on the upper edge of the pyrite band or just above the zinc mineral on the zinc-circuit tables. Later improvement in flotation operations effected the removal of the gold-bearing mineral with flotation concentrates to such an extent that the table treatment of scavenger machine tailings is now unnecessary. One table per section has been retained for treating scavenger tailings, but its function is solely that of a pilot machine in disclosing the presence of lead carbonate which at times is present in the mill feed. There is never enough carbonate to yield more than a streak of extremely fine-grained mineral, $\frac{1}{4}$ inch wide, and usually if lead carbonate is present in the mill feed the streak is not more than barely visible. The object of the pilot table is not to recover the lead but to note the presence of this mineral, as it has been found that the sulphides are appreciably oxidized in the zones where lead carbonate exists.

Both the Forrester and Hunt types of air-lift machines have been tested as cleaner units for the zinc circuit. Neither machine operated satisfactorily for this purpose on Pecos ore; it was found that if the required amount of air was furnished to maintain the heavy pulp in motion, the machines had sufficient power to float practically all the solids of the pulp. Experience with the air-lift type of machine in treating Pecos ore indicates that it is not suitable for close selective lead-zinc separations. As a machine of this type is dependent on air volume for agitation, the volume of air can not be reduced below certain limits which

are governed by the nature of the pulp being treated. These limits may be such as to produce entirely too much froth and flotation power for close selective work. As a scavenger, or where the division between floatable and nonfloatable mineral is broad, there seems to be nothing against the air-lift type of machine, and much in its favor.

Mixing and Feeding Reagents

The tabulation which follows indicates the kinds and quantities of flotation reagents used during April, 1931.

Reagent	Addition per ton of ore treated, pounds
Mica circuit:	
Cresylic acid	0.175
Lead circuit:	
Lime	.574
Potassium pentasol xanthate	.161
Zinc sulphate	1.395
Cyanide	.060
Zinc circuit:	
Lime	1.848
Copper sulphate	.783
Sodium ethyl xanthate	.500
Pine oil	.151

Lime in the form of dry lime hydrate is added to the lead and zinc circuits by slowly moving belts. The smaller and more closely controlled quantities used in the lead circuit are fed by 10-inch belts which travel at a speed of 1 foot in 2 3/4 minutes. Each belt is served by a hopper 3 feet long by 1½ feet wide by 2 feet high; the hopper tapers to a slot 3 feet long by 3 inches wide, beneath which the belt travels. Each hopper is equipped with a gate which is raised and lowered by a long threaded 5/8-inch bolt passing through a fixed nut. The gate carries a marker point which moves over a scale divided into 1/8-inch divisions. The usual rate of feeding lime into the lead-circuit pulp is from 6 to 10 pounds per hour and the amount fed is controlled by the height of the feeder gate. The usual stream of lime on the moving belt is 2¼ inches wide and ½ to ¾ inch deep. The lime within the hopper is maintained at uniform density by an arm suspended in the hopper which oscillates slowly in line with and directly over the belt. The feeder is inspected each hour.

The larger quantities of lime used in the zinc circuit are fed by long 16-inch belts which travel at a speed of 23 inches per hour. Side boards, 1 by 12 inch, are mounted over each belt so that a 12 by 12 inch trough is formed, the belt providing a moving bottom. Lime is dumped into this trough and leveled by means of scrapers which fit into the trough and are suspended by shoulders which in turn rest on the tops of the side boards. Scrapers, 1 to 12 inches deep, are available for varying the depth of lime in the trough. The lime, especially when fed in deep layers, tends to drop from the discharge end of each belt in rather large masses with consequent irregularity in feeding rates. A rotating "scratcher" is provided at the discharge end of each belt to insure uniform feeding conditions. The "scratcher" consists of a 6-inch wooden cylinder mounted on a slowly revolving shaft. Two spiral rows of 40 D spikes are driven into the cylinder: each spiral makes a half turn on the cylinder and comfortable clearance is provided between the belt and the ends of the spikes. A small

stream of water carries the lime in a launder to the point of use.

Cyanide and zinc sulphate are added as solutions by means of chain and bucket type feeders which supply syphons and decanter pipes. The solutions are made up and stored in 5 by 5 foot stock solution tanks. Each tank is equipped with a chain and bucket type feeder mounted within the tank, which discharges into the receiving end of an inverted syphon pipe; the excess solution overflows the syphon discharge and returns to the tank. Tees with ½-inch outlets arranged horizontally are placed in the horizontal run of the 2½-inch syphon line. Attached to each tee is an adjustable decanter pipe which can be swung in a vertical arc to control the quantity of solution discharged. The amount discharged depends upon the difference in elevation between the discharge of the syphon pipe and the discharge of the decanter pipe. These feeders have been found to discharge uniformly and to be accurate within 25 c.c. of solution per minute. It is probable that greater accuracy could be obtained by providing the decanter pipe with a screw adjustment but with the large quantities of dilute solutions added this refinement has not been found necessary. Two per cent cyanide solution is added at the rate of 150 to 300 c.c. per minute. Zinc sulphate solution containing 3 per cent metallic zinc is added to the lead roughers at the rate of 600 to 800 c.c. per minute and to the lead cleaners at the rate of 200 to 300 c.c. per minute. As many decanter pipes as desired may be placed in the syphon line; but for uniform results the flow through the syphon line should be considerably greater than the combined discharges of the decanter pipes and, furthermore, the syphon line should be large enough in diameter to insure a low velocity of flow through it.

Cyanide solution is made up in a mixing tank provided with a basket suspended in the top of the tank and at one side. Solid cyanide is placed in the basket, and a 1-inch centrifugal pump circulates the solution from the bottom of the tank to the basket. Enough solution to fill the 5 by 5 foot feeder stock tank can be made up and thoroughly mixed in about two hours.

Zinc sulphate solution is prepared in a tank 6 feet in diameter by 8 feet deep, which is equipped with a 30-inch Deveraux agitator. The tank is filled to the mark with water and the agitator is started. The zinc sulphate is placed in a basket as in the cyanide tank. After three to four hours agitation, the solution is transferred to the stock solution tank by a 1-inch centrifugal pump.

Copper sulphate is fed in solution through rubber hose lines leading from dissolving barrels located near the points of use. Two barrels placed in series serve each unit; water is fed to the barrels at the rate of 200 c.c. per minute by Daman feeders. The water enters at the bottoms of the barrels and rises in contact with the copper sulphate crystals. The temperature does not vary sufficiently to affect materially the amount of copper contained in the solution, which is nearly saturated.

Xanthate is added to the flotation circuit in the form of a 10 per cent solution. Most of the American-made xanthates contain variable amounts of material that is evidently xanthate or principally xanthate, which requires from 8 to 24 hours to effect solution. There is also a wide variation at times in the strength of xanthate from one drum to another. In order to obtain solutions of approximately uniform strengths at the feeders it has been found necessary to analyze the content of each drum before making up the solution and to analyze the resulting solution as well. The practice at the Pecos plant is to plan for a 12 per cent xanthate content in the first trial solution, then, after agitating for four hours, to analyze the resulting solution and dilute with water to the desired 10 per cent strength. The final solution is then run to the feeder tanks where it is handled by Daman feeders. The old-style feeders of this name are cheap, easily adjusted, and have given entire satisfaction at this plant.

When amyl xanthate, Great Western Z 5, is added to the lead circuit it has been found that only very small quantities of xanthate need be added to the zinc circuit; the zinc

circuit has indeed been operated for as long as a week without additional xanthate. This condition would seem to indicate that excess xanthate was being used in the lead circuit, yet reducing the amount added to this circuit decreases recoveries.

When pentasol xanthate, Z 6, is used in the lead circuit, the amount required is much less, as compared to amyl xanthate, but considerable additional xanthate must be added to the zinc circuit. Attempts to increase the Z 6 added to the lead circuit and thereby reduce the additional xanthate required in the zinc circuit have failed because of difficulties encountered in the functioning of the lead circuit. The pronounced difference in the actions of the two xanthates as described has been demonstrated on Pecos ore by repeated trials. When each xanthate, however, is used to best advantage there is no difference in the metallurgical results. American cyanamid reagent No. 301, which is understood to be a secondary butyl xanthate, is being used at present.

Pine oil and cresylic acid are fed to the flotation pulps by old-style Daman feeders.

DEWATERING AND HANDLING CONCENTRATES

The finished lead concentrates of each unit, after passing mechanical samplers, are mixed and fed to one 40 by 10 foot Dorr thickener. Two 40 by 10 foot Dorr thickeners are provided for finished zinc concentrate pulps, one serving each flotation unit. The zinc concentrates are also mechanically sampled before entering the thickeners. Each thickener is served by a 3-inch Dorrco simplex pump, both thickener and pump being driven by a 5-hp. motor through a speed reducer. The average motor inputs amount to 1.6 hp. and 1.04 hp. for zinc and lead thickeners, respectively.

Thickener feeds contain 28 per cent of solids in the case of the zinc pulp and from 10 to 15 per cent of solids in the lead pulps. Thickener overflow waters are practically clear and have a temperature of about 65°F. The pH values of zinc and lead thickener overflows are 11.0, and 9.5 to 10.0, respectively. Thickener rakes travel at a speed of 1 revolution in 5 minutes and 25 seconds.

Thickened pulps contain from 65 to 75 per cent of solids and are handled without difficulty by the pumps. High-pressure water is occasionally added at the tank end of the suction line in the case of unusually dense pulps. The discharges of the Dorrco pumps are 3 feet above the water level of the tanks; the altitude is practically 7,400 feet above sea level.

Each thickener product is further dewatered by an 8 by 12 foot Oliver filter equipped with a round-bottomed tank and oscillator. The filters revolve at a speed of 1 revolution in 5 minutes and 50 seconds; vacuum is maintained at 21 inches of mercury. Standard Palma twill filter cloth is used and No. 12 wire is wound over the cloth with 1¼-inch spacings. The average life of cloth is 10 months and during the latter part of this period the filter, centrifugal pump, and receiver are given a weak hydrochloric acid wash each month by circulating the acid through the unit, the pump returning the wash solution to the filter tank. The usual procedure is to dilute one-half a carboy of acid to 1 per cent strength and to circulate the solution for a period of two to three hours.

Each filter and each filtrate pump are driven by 3-hp. 1,200 r.p.m. motors which require power inputs of 1.35 hp. and 1.50 hp., respectively. The pumps are direct-connected to the motors; the filters are operated by chain drives through speed reducers.

The filtered concentrates drop directly into receiving bins located under the filters, from which they are shoveled by hand into discharge chutes located along the fronts of the bins; the chutes feed a Handy box-car loader. The lead concentrates contain from 7 to 8 per cent of moisture and the zinc concentrates from 7.5 to 9 per cent. Loading is conducted six days per week, there being storage capacity to carry over Sunday. The loading gang does

practically all the roustabout work of the plant, and is under the supervision of the warehouse clerk, since the gang's principal duties of loading concentrates and receiving and moving supplies fall directly in this department.

Filtering data for the first five months of 1931 are given in the tabulation which follows.

Filtering data, January 1 to June 1, 1931

Mill feed per day	tons	510.0
Lead concentrates produced per day	do.	52.4
Zinc concentrates produced per day	do.	124.7
Ratio of concentration in lead circuit	tons into 1	9.73
Ratio of concentration in zinc circuit, based on original mill feed	do.	4.09
Lead concentrates filtered per square foot of cloth before replacing cloth	tons	53
Zinc concentrates filtered per square foot of cloth before replacing cloth	do.	63
Lead concentrates handled per day per square foot of filter cloth	do.	0.175
Zinc concentrates handled per day per square foot of filter cloth	do.	0.208

The lead filter operates about half time and the zinc filters about 85 per cent of available time. Sufficient filter capacity is available to handle 50 per cent more concentrates with little or no change in results other than perhaps an increase of 1 per cent in the moisture content of filter cake and an increase in the speed of the pumps which serve the zinc filters.

With present amounts of concentrates handled large settling areas are provided in the Dorr thickeners, and the waste overflows are therefore practically clear. Excess thickening and filtering capacity was provided because of the possibility of increasing the concentrator capacity to 700 tons per day and also the possibility that the mill heads would contain 10 per cent of lead and 20 per cent of zinc. The present dewatering and filtering equipment has handled smoothly the concentrates resulting from the treatment of 600 tons of ore per day containing 20 per cent of zinc. The Dorr tanks would handle probably the concentrates produced by 700 tons of ore per day, but the 3-inch pumps are close to their maximum capacity at the 600-ton rate, on high-zinc heads.

The Handy box-car loader reduced the overall cost of loading concentrates as compared to hand methods of loading, from \$0.30 to \$0.19 per ton. The machine, however, does not produce a load of as uniform density as that obtained by hand shoveling, and for this reason considerable experience is required by operators in preventing over and under loading of cars. Loading lines are marked with chalk and the weight per cubic foot of material is determined several times while loading a car by weighing the contents of a box of 1 cubic foot volume. In filling this box the contents are jolted to approximate the density of material in the loaded car. After one months experience an operator is able properly to control the loading of concentrates. If wide fluctuations occur in moisture content of concentrates, even an experienced man will often be mistaken in the weight of material in a loaded car. Both the mining company and railroad, but chiefly the railroad, are interested in loading cars to as near maximum capacity as possible. As loading is conducted with this end in view, overloads are bound to occur at times. The tabulation which follows gives loading data for the first five months of 1931.

Loading data, January 1 to June 1, 1931

Total concentrates shipped	wet tons	29,468
Total maximum capacity of cars used	tons	30,886
Loading efficiency	per cent	95.4
Number of cars used		602
Average number of men per working day		9.1
Average weight handled per man	wet tons	24.65
Number of overloads		82
Number of underloads		9
Cars with satisfactory load	per cent	85

DISPOSAL OF TAILINGS

The tailings of the zinc-flotation circuits, to which the mica products and thickener overflows have been added, pass to their respective samplers. After sampling, the pulps of the two circuits join and are conveyed by the main tailings launder to Alamitos Arroyo. The latter conveys the material to the tailings pond which is formed by a dam constructed across it. The distance is about 1 mile and the average slope of the bed is about ½ inch per foot for this distance.

The dam is 325 feet long by 18 feet wide by 65 feet high near its center. It is made of earth and rock with a 6-foot puddled core along its center line and was constructed in 3-foot lifts with teams and dump carts. The front face has a slope of 2½ to 1, and the upper portion of this slope is covered with a broken rock facing; the rear face has a slope of 1½ to 1.

A 36-inch corrugated steel drain tube extends through the bottom of the dam and joins an overflow downtake at a point just in front of the upper toe of the slope. The downtake is a box, 36 inches square, supported in a wooden tower. The box is elevated from time to time as the pond fills.

Solids from the tailings pulp begin to deposit in the arroyo about 3,000 feet above the dam, and they form a rather uniformly sloping bed about 2,700 feet long to the entrance of the pond. The average exposed slope of this bed is 0.38 inch per foot; the pulp stream wanders back and forth over it, building up a deposit similar to that of a river delta. The average grade of the exposed deposit depends upon the nature of the pulp and, once established, its surface will rise and expand as the water level is raised in the pond in accordance with this grade and the surrounding contours.

The pond is roughly 300 by 300 feet in size. Shortly after entering the quiet zone of the pond the grade of bed material increases sharply, which marks the beginning of the slime deposit. As nearly as can be determined the slime material acts as a heavy liquid and its surface is level under the entire pond area.

Over a period of two years the dam settled about 6 feet near its center and this shrinkage tapered out to zero at the ends. No further movement has recently taken place and no leakage has occurred through the dam at any time.

While the mill operated with neutral and soda ash lead circuits the pond overflow water had a slight whitish turbidity. With the full lime circuit in present use the overflow water is clear and the pool area is allowed to become much smaller before raising the level.

A spillway is cut through solid rock around the dam and is of several times greater capacity than is required to handle the volume of any known run-off from cloudbursts.

The dam originally cost $31,724 and expenditures made since the original construction have brought the total cost to date to $33,000; no further expenditures on it are contemplated. It will retain readily 800,000 tons of material and still act as a clarifier. On the

basis of a concentration ratio of 100 to 30, the dam will serve to store the tailings from the treatment of 1,000,000 tons of ore; this gives an estimated cost of \$0.03 per ton of mill heads for the storage of tailings.

WATER SUPPLY

The mill uses about 1,200 gallons of water per ton of ore treated. No water is reclaimed either from the mill or tailings pond, except a part of the lead-thickener overflow, which is reused as wash water on the tables. There is a plentiful supply of fresh water which is pumped from the river at a cost of \$32 per million gallons; this is as low as the cost of returning reclaimed water to the mill. Another reason for not using reclaimed water is the fact that the flotation of mica requires entirely fresh water. The effect of returning used water to the mica-flotation circuit was tested by returning the mica cleaner tailings pulp to the classifier serving the grinding unit. Although the returned pulp did not contain any reagent other than cresylic acid, the latter caused serious flotation of sulphides with the mica froth.

SAMPLING

All mechanical samplers used are of a type developed at the plant. The design is based on the principle of a slot, uniform in width, crossing the pulp stream in a straight line and at a uniform velocity. The head sampler is operated by a Stromberg time clock to cut the stream at 20-minute intervals; the motor reversing mechanism is a 3-pole double throw switch. The latter is thrown by the sampler mechanism near the end of the cutter stroke and after the operating current is cut off. All other samplers are controlled by the operation of the head sampler through 440-volt lines. The time clock is placed in the mill office and operates a 110-volt relay, which in turn operates the current of the 440-volt line on the head sampler. The weight of heads is calculated from the weight of sample taken; after small adjustments to the factor are made, based on comparisons with shipments, the results obtained are as accurate as can be expected from such apparatus and are fully comparable with tonnages based on bin measurements. The concentrate and tailings samplers are not calibrated for tonnage determinations, as the weights of these products are computed by the Hoover bimetallic formula from assays and the feed tonnage.

The sample pulps are received in canvas sacks fitting closely into iron tubs which are connected with the vacuum system. The pulps are filtered shortly before the end of the shift, at which time they are sent to driers and fresh sacks substituted.

BLOWERS AND VACUUM PUMPS

Air for the Minerals Separation machines is supplied at 12 to 14 ounces pressure by one No. 2 Roots blower, which is driven at a speed of 317 r.p.m. by a 10-hp. motor through a Texrope drive. The blower has a displacement of 1,521 cubic feet per minute; the motor input is 6.9 hp. A second blower is held in reserve, but during the 4½ years of operation to date it has been needed only once, while changing bearings on the operating machine.

A No. 2½ Roots blower supplies air for the air-lift type machines at 2 to 2½ pounds pressure. It is driven at a speed of 235 r.p.m. by motor which requires a power input of 17.3 hp. The blower displacement amounts to 1,668 cubic feet per minute, which is sufficient for about 40 linear feet of machine.

Air for blowing the cake and canvas on the filters is furnished by one No. 3 Roots blower at 7 pounds pressure. It was installed to supply air for the pulp conditioners as

well as the filters. It is driven at a speed of 387 r.p.m. by a 50-hp. motor through a Link-Belt chain drive; power input is 17.7 hp. The displacement amounts to 900 cubic feet of air per minute; the filters require about 300 cubic feet of free air per minute. The 50-hp. motor was selected for two reasons: First, there was the possibility that full blower speed of 425 r.p.m. and air at 10 pounds pressure might be needed; these requirements would mean nearly full load for the motor; second, the 50-hp. motor corresponds in size and speed with those used to drive vacuum pump, flotation machinery, and other machinery, and its selection assisted in standardizing motors and parts.

Two vacuum pumps, one a spare, are installed for the filters. They are 26 by 10 inch Ingersoll Rand machines and are driven at a speed of 222 r.p.m. by 50-hp. motors through Lenox short-belt drives. The displacement of each is 1,356 cubic feet per minute; the power input is 28.8 hp. The reasons for the selection of the size of motor used are the same as given for the motor operating the No. 3 Roots blower and, in addition, the idle 50-hp. motor is considered as an emergency spare motor for the operation of other machines used in the mill.

GENERAL MILL POWER FACTOR AND POWER CLASSIFICATIONS

Two sets of 2,200-volt feeders connect the mill and power house; the mill loads are so grouped as to divide the entire load about evenly between the two sets. The power factor on the No. 1 line is 0.860 and on the No. 2 line, 0.854. All current used in or about the mill and offices, including that for lighting, is carried on these lines.

The distribution of power used in the mill follows:

Voltage	Number of motors running	Total rating, hp.	Total input, hp.	Per cent of full load
2,200	8	600.0	509.5	84.9
400	79	377.5	243.5	64.5
Total	87	977.5	753.0	77.0

Of the total power input, 67.66 per cent is through the 2,200-volt motors and 32.34 per cent through the 440-volt motors; 40.1 per cent of the total input is through 2,200 volt synchronous motors with unity power factor design.

METALLURGY

Tables 1, 2, 3, and 4 present assay-screen analyses of mill heads, lead concentrates, zinc concentrates and tailings, respectively, for April, 1931. Metallurgical results for 1930 are given in Table 5.

At Pecos it is the custom to make daily economic reports based on metal prices, smelter contracts, freight charges, and concentrate loading costs of that particular day. Table 6 presents an economic report for 1930, using metal prices and smelter and handling charges as of March, 1927. The economic result of the Pecos mill operation is expressed as the per cent of the gross heads returned by the smelter settlement for the concentrates, including deductions for freight and loading. The gross value of the mill heads is computed from the entire gold, silver, lead, copper, and zinc contents and the full market prices of these

metals. The gross value of the lead-circuit heads is taken as the combined gross values of the gold, silver, lead, and copper contents. As metal prices, loading costs, and smelter contracts are variable, two economic reports are made, one based on the metal prices and concentrate charges of March, 1927, and the other based on current metal prices and charges. The report based on March, 1927, prices is used to compare metallurgical results, since it is based on a fixed set of costs and metal prices and may be considered a metallurgical barometer. The second report, based on current prices and charges, may be considered as a financial barometer. In Table 6 metal prices and charges as of March, 1927, have been used.

Pecos ore contains appreciable amounts of gold, silver, and copper, the combined value of which renders it impossible to base metallurgical results on the recoveries of lead and zinc alone. Table 7 gives gold, silver and copper contents of mill heads each year from 1927 to 1930 and also indicates the recoveries of these metals in the lead concentrates. It may be noted in Table 7 that although the contents of these metals have decreased in mill heads since 1927, considerably improved results have been obtained in concentrating these metals with the lead.

Table 8 presents a summary of milling costs for 1930; Table 9 gives the distribution of power for 1930.

Table 1.--Screen-assay analysis of mill heads, April, 1931

Screen size, mesh	Weight, per cent		Assays							Per cent of total[1]						
			Ounces per ton		Per cent											
	Individual	Cumulative	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble
Heads by sample	-	-	0.11	3.64	4.82	0.98	16.40	11.40	33.1	-	-	-	-	-	-	-
Heads by products	-	-	.124	3.72	4.78	.97	16.43	13.00	32.4	-	-	-	-	-	-	-
Plus 65	5.13	5.13	.07	.81	0.56	.17	1.60	4.60	79.1	2.90	1.12	0.61	0.90	0.50	1.85	12.54
Plus 100	4.23	9.36	.07	.98	.70	.25	4.30	3.90	64.4	2.39	1.12	.63	1.09	1.11	1.23	8.40
Plus 150	8.66	18.02	.07	1.63	1.02	.64	13.00	7.10	48.7	4.90	3.80	1.84	5.73	6.86	4.70	13.03
Plus 200	6.11	24.13	.08	1.90	1.06	.96	18.90	13.70	30.8	3.95	3.12	1.36	6.07	7.03	6.47	5.81
Minus 200	75.87	100.00	.14	4.46	6.02	1.10	18.30	14.70	25.7	85.83	90.84	95.56	86.21	84.50	85.75	60.22

1 - Based on products assays.

Table 2.--Screen-assay analysis of lead concentrates, April, 1931

Screen size, mesh	Weight, per cent		Assays							Per cent of total[1]						
			Ounces per ton		Per cent											
	Individual	Cumulative	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble
Heads by sample	-	-	0.88	23.37	38.30	4.49	13.90	12.20	3.5	-	-	-	-	-	-	-
Heads by products	-	-	.90	23.55	38.76	4.56	14.06	11.89	3.5	-	-	-	-	-	-	-
Plus 150	1.61	1.61	.34	7.80	8.20	2.31	23.40	11.50	22.0	0.61	0.53	0.34	0.81	2.68	1.60	10.06
Plus 200	6.80	8.41	1.07	12.07	10.50	3.13	23.40	19.80	5.6	8.06	3.48	1.84	4.66	11.32	11.35	10.92
Minus 200	91.59	100.00	.90	24.68	41.40	4.70	13.20	11.30	3.0	91.33	95.99	97.82	94.53	86.00	87.05	79.02

1 - Based on products assays.

Table 3.--Screen-assay analysis of zinc concentrates, April, 1931

Screen size, mesh	Weight, per cent		Assays							Per cent of total[1]						
			Ounces per ton		Per cent											
	Individual	Cumulative	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble
Heads by sample	-	-	0.03	3.05	1.54	0.99	54.20	6.70	2.3	-	-	-	-	-	-	-
Heads by products	-	-	.03	3.01	1.52	.99	54.27	6.61	2.4	-	-	-	-	-	-	-
Plus 150	8.90	8.90	.01	1.70	.90	1.04	51.90	6.50	5.9	3.03	5.04	5.26	9.44	8.51	8.76	21.63
Plus 200	11.33	20.23	.04	2.60	.90	1.18	53.80	7.50	2.0	15.44	9.80	6.70	13.60	11.24	12.87	9.35
Minus 200	79.77	100.00	.03	3.21	1.68	.95	54.60	6.50	2.1	81.53	85.16	88.04	76.96	80.25	78.37	69.02

1 - Based on products assays.

Table 4.--Screen-assay analysis of tailings, April, 1931

Screen size, mesh	Weight, per cent		Assays							Per cent of total[1]						
			Ounces per ton		Per cent											
	Individual	Cumulative	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble
Heads by sample	-	-	0.03	1.05	0.80	0.37	1.60	14.10	44.8	-	-	-	-	-	-	-
Heads by products	-	-	.03	1.08	.82	.38	1.61	13.96	-	-	-	-	-	-	-	-
Plus 65	11.80	11.80	.04	.76	.40	.16	1.16	5.40	63.4	13.96	8.28	5.74	5.04	8.51	4.57	15.29
Plus 100	7.31	19.11	.06	.92	.56	.25	1.66	4.60	65.3	12.98	6.21	5.01	4.77	7.51	2.41	9.76
Plus 150	7.68	26.79	.05	1.01	.64	.48	1.84	9.10	58.4	11.37	7.16	5.99	9.82	8.77	5.01	9.18
Plus 200	10.20	36.99	.05	.91	.50	.56	1.36	17.30	44.7	15.09	8.57	6.23	15.12	8.64	12.64	9.33
Minus 200	63.01	100.00	.025	1.20	1.00	.39	1.70	16.70	43.8	46.60	69.78	77.03	65.25	66.56	75.37	56.44

1 - Based on products assays.

Table 5. -Metallurgical results for the year 1930

	Weight, tons	Assays								
		Ounces per ton		Per cent						
		Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble	Silica	Sulphur
Heads	151,943.0	0.107	3.37	4.93	0.83	15.41	11.40	32.10	-	18.31
Lead concentrates	[1]16,244.0	0.768	19.98	37.81	4.02	13.47	11.47	7.27	4 50	24.31
Zinc concentrates	[1]36,406.6	.036	3.05	1.63	1.00	54.45	6.99	2.32	1.60	32.18
Tailings	[2]99,292 4	.024	.77	76	.26	1.33	14 20	48.30	-	13.04
Totals	-	-	-	-	-	-	-	-	-	-

	Weights of metals							Distribution of metals, per cent[3]				
	Ounces		Pounds									
	Gold	Silver	Lead	Copper	Zinc	Iron	Insoluble	Gold	Silver	Lead	Copper	Zinc
Heads	16,227.1	511,582	14,984,733	2,510,710	46,837,814	-	-	-	-	-	-	-
Lead concentrates	12,477.1	324,499	12,283,715	1,306,075	4,377,417	3,727,789	2,363,471	76.89	63.43	81.97	52.02	9.35
Zinc concentrates	1,306.7	111,194	1,184,565	727,969	39,645,584	5,091,745	1,688,518	8.06	21.73	7.91	28.99	84.64
Tailings	2,430.7	76,889	1,515,884	520,549	2,631,914	-	-	15.05	14.84	10.12	18.99	6.01
Totals	16,214.5	512,582	14,984,164	2,554,593	46,654,915	-	-	100.00	100.00	100.00	100.00	100.00

1 - Calculated.

2 - By difference.

3 - Based on metal accounted for in concentrates and tailings.

Table 6.--Economic report for the year 1930[1]

	Lead circuit	Zinc circuit	Total
Gross value per ton of heads	$13.40	$20.62	$34.02
Smelter returns for concentrates, per ton of heads	$ 7.09	$ 7.64	$14.73
Per cent of gross value of heads in smelter returns	52.91	37.05	43.30

1 - Metal prices, freight and treatment costs for March, 1927, used.

Table 7.--Yearly comparison of gold, silver, and copper recovered in lead concentrates, 1927 to 1930, inclusive

Year	Heads assays			Recoveries of metals in lead concentrates					
				Weights per ton of heads treated			Per cent		
	Gold, ounces per ton	Silver, ounces per ton	Copper, per cent	Gold, ounces	Silver, ounces	Copper, pounds	Gold	Silver	Copper
1927	0.124	4.16	1.04	0.049	2.18	8.85	39.88	52.52	42.57
1928	.106	3.39	1.00	.063	1.80	7.71	59.26	53.09	38.55
1929	.100	2.81	1.02	.069	1.63	9.87	68.74	57.98	48.38
1930	.107	3.37	.83	.082	2.14	8.63	76.89	63.43	52.02

Table 8.—Summary of milling costs for year 1930

	Cost per ton of ore treated				
	Labor	Supplies	Miscellaneous	Power[1]	Total
Superintendence	$0.0888	$0.0007	–	–	$0.0895
Grinding	.0500	.1546	–	$0.0670	.2716
Flotation	.1157	.0374	–	.0723	.2254
Reagents	–	.2946	–	–	.2946
Tabling	.0226	.0028	–	.0128	.0382
Sampling	.0124	.0022	–	.0059	.0205
Thickening	.0001	.0008	–	.0016	.0025
Water	.0003	.0001	$0.0431	.0102	.0537
Filtering	.0201	.0027	–	.0145	.0373
Heating	.0001	.0070	–	–	.0071
Disposal of tailings	.0002	.0006	–	–	.0008
Lighting	–	.0010	.0035	–	.0045
Lubrication	–	.0040	–	–	.0040
Mill clean-up	.0170	.0010	–	–	.0180
Miscellaneous	.0003	.0010	–	–	.0013
Subtotal	0.3276	0.5105	0.0466	0.1843	1.0690
Experimental	0.0027	0.0027	0.0266	–	0.0320
Laboratory	–	–	0.0595	–	.0595
Royalties	–	–	0.0293	–	.0293
Total	0.3303	0.5132	0.1620	0.1843	1.1898

1 - Power costs about $0.006 per kilowatt-hour at the motors.

Table 9.—Power details and distribution for the year 1930

	Average daily consumption, kilowatt-hours	Average load, horsepower	Kilowatt-hours per ton of ore treated	Power cost per ton of ore treated	Per cent of total power used
Grinding	4,603	257.0	10.96	$0.0677	35.08
Flotation	4,924	274.9	11.73	0.0724	37.53
Tabling	874	48.8	2.08	0.0130	6.66
Thickening	111	6.2	0.26	0.0016	0.85
Filtering	983	54.8	2.34	0.0144	7.49
Sampling	398	22.2	0.95	0.0058	3.03
Laboratory	156	8.7	0.37	0.0023	1.19
Mill water	694	38.7	1.65	0.0102	5.29
Mill lighting	236	13.2	0.56	0.0034	1.80
Camp lighting	142	7.9	0.34	0.0021	1.08
Totals	13,121	732.4	31.24	0.1929	100.00

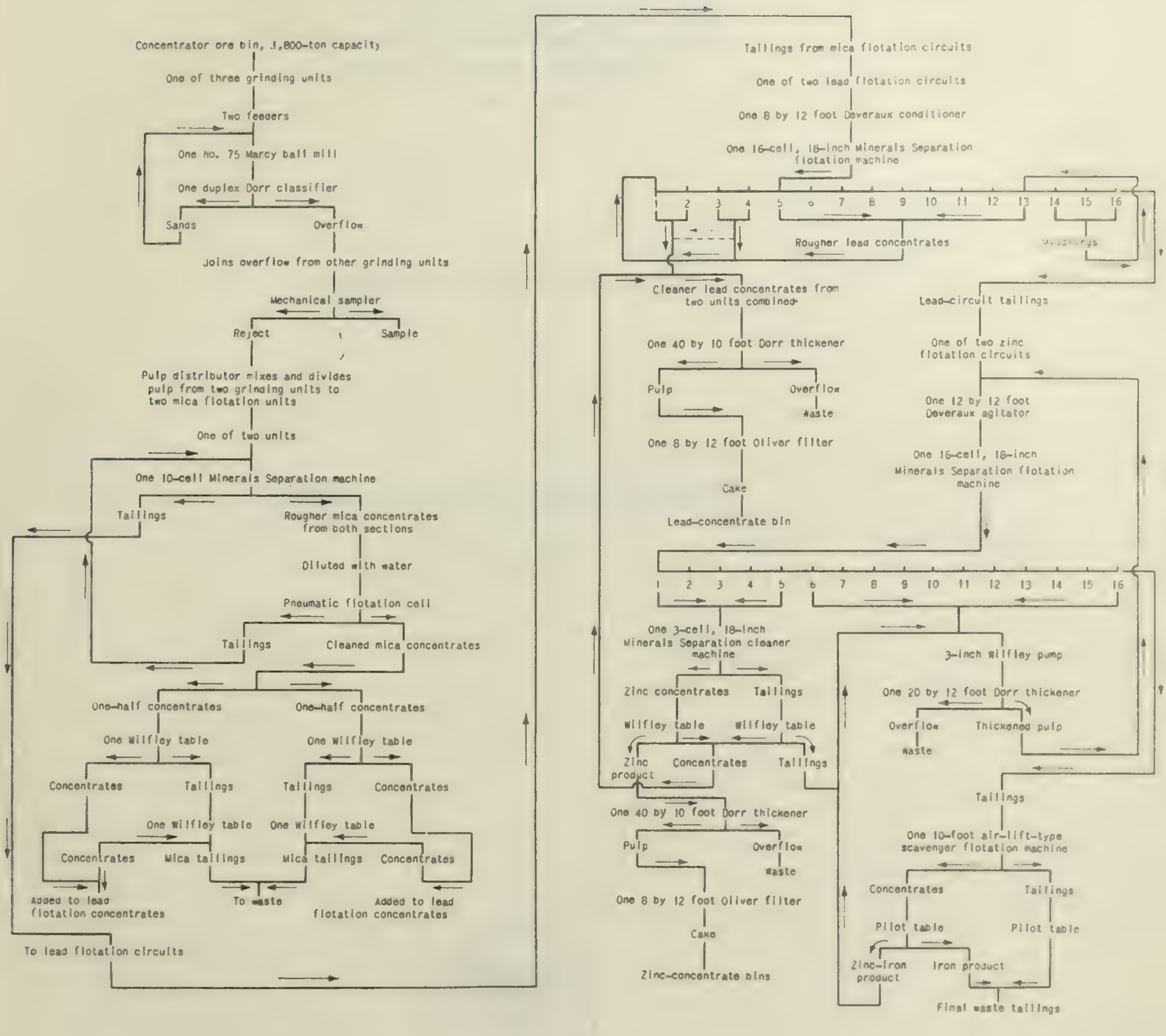

Figure 1.- Flow sheet of concentrator

I. C. 6606 March, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXXV

BY

Frederick W. Lee

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]
No. 35
Compiled by Frederick W. Lee[2]

List of contributing editors of Geophysical Abstracts:

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Ayvazoglou, W., U. S. Bureau of Mines, Department of Commerce, Washington, D. C.
Barton, Dr. D. C., Petroleum Building, Houston, Tex.
Belluigi, Dr. Arnaldo, Corso Vittorio Emanuele 178, Parma, Italy.
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Petrowsky, A., Wasilly Ostrov, 21 Linia No. 8-A, Leningrad, U.S.S.R.
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Zuschlag, Dr. Theodor, 229 Rutland Ave., West Englewood, N. J.

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2 - Senior physicist, U. S. Bureau of Mines.

1. GRAVITATIONAL METHODS

(648) DETERMINATION OF "G" BY MEANS OF THE FREE-SWINGING PENDULUM

By C. H. Swick

Bulletin of the National Research Council, Washington, D. C., No. 78, 1931, pp. 151-166.

After a brief historical review on the discovery of the pendulum and the underlying principles which made possible its use for gravity measurements, the author mentions the absolute determination of gravity by the pendulum constructed by Henry Kater and the improved types of the reversible pendulum developed by Repsold. For the determinations made by the relative method two general types of modern gravity apparatus are described: (1) The older, known as the von Sterneck apparatus (1880), and (2) that known as the Mendenhall, or Coast and Geodetic Survey apparatus, designed in 1890. The latter is simpler and easier to adjust and use; its accuracy is about 0.001 or 0.002 cm. per second^2. Various sources of errors are enumerated. The chief differences between the two apparatus are noted.

A list of books and articles for a more detailed study of the different kinds of pendulum apparatus and methods used in determining gravity is added. --W. Ayvazoglou.

(649) GRAVITY MEASUREMENTS WITH THE EÖTVÖS TORSION BALANCE

By Donald C. Barton

Bulletin of the National Research Council, Washington, D. C., No. 78, 1931, pp. 167-190.

The principles of the Eötvös torsion balance and the theory of its action are described. The distortion of the level surfaces caused by the introduction of a heavy mass causes the rotation of the beam. The curvature and gradient can be calculated from the values of the torque observed at the stations. The effects of topography and hidden irregularities of mass in the soil and subsoil close to the instrument can be avoided by proper choice of the station site, or eliminated to a considerable extent by taking measurements at additional stations within a short distance of the station. Under the last two headings the author discusses the units of measurement and variation of gravity.

References written by Baron Eötvös, as well as a series of papers written in English, are added to the article.--W. Ayvazoglou.

(650) GRAVITY DISTRIBUTION

By A. Belluigi

Gerlands Beiträge zur Geophysik, Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 3, 1931, pp. 227-240.

Analytical and graphical investigations of the distribution of gravity in a deep lying, assymmetrical anticline are discussed.

The errors that may occur in considering relief and gravity and that are due to confusing the effects of variations in the surroundings with anomalous concentration (that is, mineral deposits) are investigated.--W.A.R. Reprinted from Science Abstracts, Sect. A, vol. 34, No. 402, 1931 (469).

(651) MAKE GEOPHYSICAL SURVEYS IN EASTERN MISSISSIPPI

Editorial note

The Oil Weekly, Houston, vol. 62, No. 13, 1931, p. 60.

Considerable geophysical work in Noxubee and Winston Counties, in east central Mississippi, carried on by the Torsion Balance Exploration Co. is reported.

Tests wells are now being drilled, one in Winston County and one in Noxubee County. A party of the Geophysical Research Corporation of Houston has been working recently in various parts of Mississippi.--W. Ayvazoglou.

(652) BRAZORIA COUNTY WILDCAT SHOWS OIL

By Neil Williams

The Oil and Gas Journal, Tulsa, Okla., vol. 30, No. 16, 1931, p. 43.

Williams gives an account of the interest centered on Manvel geophysical salt-dome prospect in northern Brazoria County, discovered in 1929 by torsion balance.--W. Ayvazoglou.

2. MAGNETIC METHODS

(653) PROGRESS OF WORK IN TERRESTRIAL MAGNETISM OF THE UNITED STATES COAST AND GEODETIC SURVEY, JULY 1, 1927, TO JUNE 30, 1930

By D. L. Hazard

Union géodesique et géophysique internationale, Section de magnétisme et électricité terrestres, Bull. 8, 1931, pp. 76 (I-VII).

Geophysicists employing magnetic methods of prospecting are more and more making use of the results of the work on terrestrial magnetism carried out by the U. S. Coast and Geodetic Survey, which keeps in close touch with the

progress of work of the American Geophysical Union and other scientific organizations.

In this article the author gives a brief review of magnetic surveys carried on in the United States, Alaska, Hawaii, and the Philippine Islands. The work of the magnetic observatories, attention given to the improvement of the instrumental equipment of the observatories, and computations made by them are briefly described. A list of publications and articles issued by the U. S. Coast and Geodetic Survey on this subject is added.--W. Ayvazoglou.

(654) REPORT ON THE MAGNETIC INVESTIGATIONS OF THE GEOPHYSICAL INSTITUTE OF LWOW UNIVERSITY DURING THE YEARS 1928-1929

By Edward Stenz and Henryk Orkisz

Kosmos, Lwow, vol. 55, No. 3-4, 1930, pp. 429-443.

Magnetic research work at the Geophysical Institute of Lwow University was initiated in 1928. A detailed magnetic survey of southern Poland to discover local anomalies in the Sub-Carpathian oil region has been carried on since that time.

In this article the authors describe the following work:

1. Investigations in the vicinity of Lwow during the year 1928. Two thousand two hundred and fifty square kilometers was surveyed, the average distance between the stations being 3 kilometers and the number of them 242. Two hundred and fifty measurements were made with a Chasselon inclinometer. The results of these measurements were published by the same authors in 1929, Kosmos, vol. 54, No. 1-2 (see Geophys. Abs. 11).

2. The absolute measurements made during the year 192o. An area of 4,050 square kilometers was surveyed. Two hundred and fifty-one observations of inclination with stations 4 kilometers apart were made. Moreover, on an area of 2,025 square kilometers the horizontal intensity was measured at 65 stations and the declination at 35 stations. A map of inclination in the vicinity of Strij and a map showing the anomalies of inclination in the vicinity of Lwow and Strij are given.

3. Measurements with a Schmidt balance during the year 1929. The 990 points of observation were 1 kilometer apart. A map of horizontal intensity in the vicinity of Drohobycz and Skole is given

4. Remarks concerning the diurnal and annual variations of magnetic elements in Daszawa. The records of the four magnetic elements, D, H, Z, and I, in monthly means for each hour, are given in a table and are graphically represented in four figures.

5. The magnetic work during 1930. Measurements in the Carpathians and the lowland in front of them were carried on. The vertical component

for prospecting purposes was to be measured between the rivers Swica and Lomnica on an area of 1,300 square kilometers.--W. Ayvazoglou.

(655) VARIATIONS SÉCULAIRE DE LA DISTRIBUTION DES ÉLÉMENTS MAGNÉTIQUES

(SECULAR VARIATIONS OF THE DISTRIBUTION OF MAGNETIC ELEMENTS)

By L. Éblé

Annales de l'Institut de Physique de Globe de l'Université de Paris, Paris, vol. 9, 1931, pp. 87-95.

The author discusses how the determination of the secular variations of the magnetic elements can be expressed as a function of time, thus making it possible to calculate these elements for any epoch desired. The usefulness of determining the values of the magnetic elements for a datum chosen in advance for tracing corresponding maps is mentioned especially. Based on these considerations, Éblé gives a map of declination in France for the epoch 1931, 0.--W. Ayvazoglou.

(656) EFFECT OF SUNSPOT CYCLE ON MAGNETIC DIURNAL VARIATION AT APIA

By C. J. Westland

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, Md., vol. 36, No. 1, 1931, pp. 6-8.

In this article the data used are the hourly values in magnetic declination, D, and horizontal intensity, H, each hourly value having been computed from the mean ordinate to the curve during the 60 minutes commencing at the Greenwich hour stated. The difference between the highest and lowest hourly value in the day is called simply the "range." Results of harmonic analyses are given, in accordance with the notation:

H or $D = m + r_1 \sin (A_1 + t) + r_2 \sin (A_2 + 2t) + \ldots\ldots$ --Author's abstract.

(657) MAGNETIC MAP OF TRI-STATE COMPLETED

By M. D. Harbaugh

Mining and Metallurgy, New York, vol. 13, No. 302, 1932, p. 86.

A magnetometer survey of the Tri-State mining district has recently been completed by the Missouri Bureau of Geology and Mines in cooperation with the Tri-State Zinc and Lead Ore Producers' Association. Jasper, Newton, and most of Lawrence Counties, Mo.; Cherokee County, Kans.; and Ottawa County, Okla., are included in the map.

Observations of vertical magnetic intensity were made with an Askania balance, and all readings were reduced to an assumed datum tied to the U. S. Magnetic station at Mt. Vernon, Mo.

The purpose of the survey was to see what, if any, relationship might be revealed between the geology and location of the many mining fields in the district and the magnetic anomalies in the earth's field. The map does reveal very marked anomalies, some of which obviously reflect known geological conditions, especially faulting.

Inquiries regarding the magnetic map may be directed to the secretary of the Tri-State Zinc and Lead Ore Producers' Association, Box 95, Miami, Okla.--W. Ayvazoglou.

3. SEISMIC METHODS

(658) GEOPHYSICAL MEETING ON 1930, NOVEMBER 7

Editorial note.

The Observatory, London, vol. 54, No. 680, 1931, pp. 15-18.

This is the report of a meeting for the discussion of geophysical subjects held on November 7, 1930.

The meeting was opened by a discussion on microseisms. An account of the work of S. K. Bannerji on this subject was given by Sir Gilbert Walker. Some examples of the development of microseisms, taken from Bannerji's work, were discussed; the latter considered that the microseisms were generated at the sea bottom and had their origin in variations of pressure which took place there in connection with storms.

An account of an examination of the Kew seismological traces was given by A. W. Lee, who considered that some of the larger oscillations registered at Kew undoubtedly were due to the shaking of the observatory by wind.

Dr. Jeffreys gave an account on two papers, one dealing with the problem of the damping of a wave set up by a simple shock consisting of a sudden displacement and the second with the "Revision of seismological tables."--W. Ayvazoglou.

(659) COMPARISON OF TWO METHODS FOR INTERPRETATION OF SEISMIC TIME-DISTANCE GRAPHS WHICH ARE SMOOTH CURVES

By Maurice Ewing and L. Don Leet

Transactions American Institute Mining and Metallurgical Engineers, Geophysical Prospecting, 1932, pp. 263-270.

The only time-distance graphs considered in this article are those obtained in localities where the velocity of propagation depends solely on the depth beneath the surface; that is, where there is no slope in the subsurface. The authors' summary reads as follows:

1. Smoothly curved time-distance graphs actually are obtained, notably in the Gulf coastal plain sediments.

2. Although it is legitimate to approximate these smooth curves by a number of straight lines for the purposes of calculating depth at a given point or points, it is a serious error to attribute physical significance to the arbitrary set of lines used.

3. The formula, given in this report, for the interpretation of curved time-distance graphs without assuming a law of velocity increase with depth, is easy to apply and gives far more useful and satisfactory results.

The derivation of the formula for treating curved time-distance graphs is given.--W. Ayvazoglou.

(660) LA PROSPÉCTION SEISMIQUE DU SOUS-SOL

(SEISMIC PROSPECTION OF THE SUBSOIL)

By Raymond Maillet and Jean Bazerque

Annales des Mines, Paris, vol. 20, No. 10, 1931, pp. 287-341.

The article is divided into three parts: The first part deals with the technical principles of seismic prospection, the second with the instruments, and the third with examples of the application on the terrain. Contents are as follows:

Part 1, Technical principles. (1) General remarks; (2) Horizontal layers; (3) Movable layers lying between two firmer ones; (4) Inclined layers; (5) Other structures; (6) The point of folding formed by two straight lines; (7) Bent branches; (8) Abacuses; (9) Vanishing of deep waves; (10) Ambronn's and Schweydar's curves.

Part 2, Description of instruments. (1) General characteristics; (2) Seismographs; (3) Measuring bridges; (4) Electrical adjustment of seismographs; (5) Recording; (6) Recording of the explosion; (7) Time meters; (8) Unrolling of the film; (9) Arrangement of the instruments at the central registration station.

Part 3, Examples of application on the terrain. (1) Investigation of the foundation layer; (2) Homogeneous terrain; (3) Salt dome.

Twenty-nine figures illustrate the article.--W. Ayvazoglou.

(661) PROPAGATION OF LOVE WAVES

By S. Higuchi

Tohoku University Scientific Reports, Tokyo, vol. 19, No. 6, 1930, pp. 793-800.

Investigates the problem of the propagation of a Love wave along some complex superficial layers of the earth. With the use of rather simple calculations, the conditions of the existence of the wave and of no reflected wave in the regions of the incident wave, etc., are derived.--Author's abstract.

4. ELECTRICAL METHODS

(662) INVESTIGATIONS MADE IN COOPERATION WITH RADIORE COMPANY OF CANADA, LIMITED, SCHLUMBERGER ELECTRICAL PROSPECTING METHODS, AND SWEDISH AMERICAN PROSPECTING COMPANY OF CANADA

By L. Gilchrist and J. B. Mawdsley

Canada Department of Mines, Geological Survey, Memoir 165, 1931, pp. 1-77.

The geophysical exploration companies mentioned above were invited to employ their various methods on one and the same proving ground (the Abana mine) in order to form a just estimate of their possibilities. This article contains seven chapters:

Chapter I. Introduction, by J. B. Mawdsley.

Chapter II. Elementary principles of magnetic, electrical, and electromagnetic methods of prospecting (magnetic methods of prospecting, electrical and electromagnetic methods of prospecting, and fundamental mathematical formulas), by L. Gilchrist.

Chapter III. Geological conditions at the site of the investigation (Abana mines, Quebec), by J. B. Mawdsley.

Chapter IV. Electrical and magnetic conditions at the site of investigation (conditions recognizable by a preliminary examination, and conditions revealed during mining operations on the 300-foot level), by L. Gilchrist.

Chapter V. Magnetic survey of Abana property, by L. Gilchrist.

Chapter VI. Electrical and electromagnetic surveys of Abana property (method of procedure; explorations carried out by the field party of the Radiore Co. of Canada (Ltd.); explorations carried out by the field party of the Schlumberger Electrical Prospecting Methods; explorations carried out by the field party of the Swedish American Prospecting Co. of Canada (Ltd.), by L. Gilchrist and J. B. Mawdsley.

This report is, according to the authors, an attempt to help all who are interested to understand better the field work of prospecting practice. It is emphasized in the report that only mineral deposits which are distinctly magnetic or electrically conductive can be located by these methods of exploration in their present state of development.

The authors conclude that the geological investigation and the magnetic and electrical surveys carried out over the Abana mineral deposits show that the physical conditions existing there are complex but that, despite this complexity, the magnetic and electrical methods of exploration, when used intelligently, are feasible and productive of valuable results.--W. Ayvazoglou.

(663) GEOPHYSICAL INVESTIGATIONS AT THE MAMMOTH CAVES, KENTUCKY, AND IN SUDBURY BASIN DISTRICT, ONTARIO

By A. S. Eve, D. A. Keys, H. G. I. Watson, J. H. Swartz, and J. B. Mawdsley.

Canada Department of Mines, Geological Survey, Memoir 165, 1931, pp. 78-160.

This article forms the second part of Memoir 165. The following items are discussed in its three chapters:

Chapter I. Introduction. Places of carrying out the work and members of the parties in the field are enumerated.

Chapter II. Geological accounts:

1. Geology of the Mammoth Cave region, by J. H. Swartz.

2. Geology of part of the Falconbridge and Errington properties in the vicinity of Sudbury, Ontario, by J. B. Mawdsley.

Chapter III. Investigations in applied geophysics:

1. Radio and electromagnetic waves at Mammoth Cave, Ky., by A. S. Eve and D. A. Keys.

2. Earth resistance methods.

3. The self-potential method.

4. Magnetic surveys.

5. The Bieler-Watson method, by H. G. I. Watson.

Twenty-four tables and 32 figures illustrate the article.--W. Ayvazoglou.

(664) EXPERIMENTS IN ELECTRICAL EXPLORATION MADE IN THE SUMMER OF 1929

By J. Gilchrist

Canada Department of Mines, Geological Survey, Memoir 165,
1931, pp. 161-189.

This report is concerned largely with work carried out at the Abana mine in northwestern Quebec. Some work carried out at the Falconbridge and Errington mines, Sudbury district, Ontario, is discussed because of its relation to similar work at the Abana mine.

The results of the following investigations are presented:

I. Electrical investigations at the Abana mine.

(a) Measurements of resistivity by Wenner-Lee method with a megger.

(b) Measurements of resistivity by the central electrode method (a central current electrode and two end current electrodes) with a megger.

II. Electrical investigations at the Falconbridge mine. Measurements of resistivity by the central electrode method with (1) a megger and (2) a milliammeter and potentiometer.

III. Electrical investigations at the Errington or Treadwell-Yukon mine. Measurements of resistivity by the central electrode method with a milliammeter and potentiometer.

IV. Magnetic investigations at the Falconbridge mine and the Errington mine. A magnetometer exploration with the Tiberg magnetometer.—W. Ayvazoglou.

(665) ELECTRICAL PROSPECTING

By A. Belluigi

Gerlands Beiträge zur Geophysik, Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 1, No. 3, 1931, pp. 241-254.

In calculating the depths of mineral deposits from the results of electrical field observations the chief difficulty is the determination of the lateral dimensions of the deposit.

Approximate solutions can be obtained. Use is made of the relation between the potential and electromagnetic fields, especially fields with a nonelliptical structure. The Biot-Savart formula can be used to calculate the depth of the deposit.—W. A. R. Reprinted from Science Abstracts, Sec. A, vol. 34, No. 402, 1931, p. 469.

(666) A UNIFORM EXPRESSION FOR RESISTIVITY

By Sherwin F. Kelly

Transactions American Institute Mining and Metallurgical Engineers, Geophysical Prospecting, 1932, pp. 141-143.

In this article Kelly discusses the need for geophysicists to adopt a uniform mode of expressing the electrical resistivity of geological formations. He urges the adoption henceforth, save in exceptional cases, of a standard expression for resistivity.

Examination is made of expressions used by various authors, such as ohms per meter per meter square, ohms per meter meter square, ohms per centimeter per square centimeter, ohms cm. cm.^{2}. In the opinion of the author the last expression provides the clue to a desirable simplification. Writing it out thus: $\frac{\text{ohms}}{\text{cm. cm.}}\,2$, and simplifying, ohms x $\frac{\text{cm.}^2}{\text{cm.}}$, the final expression becomes ohms x centimeters.

Resistivities, therefore, can be expressed as ohm-centimeters, ohm-meters, etc. The author concludes: "Therefore, because of its simplicity, the linear relation it emphasizes, and the easily handled figures that result from its use, the recommendation is urgently made that henceforth geophysicists adopt the expression and unit 'ohm-meters' when speaking of electrical resistivities".

This form is already in use by the U. S. Bureau of Standards as the unit in its work on soil resistivity.--W. Ayvazoglou.

(667) EFFECT OF IMPREGNATIC WATERS ON ELECTRICAL CONDUCTIVITY OF SOILS AND ROCKS

By Karl Sundberg

Transactions American Institute Mining and Metallurgical Engineers, Geophysical Prospecting, 1932, pp. 367-391.

This paper discusses primarily the main factors that determine the electrical conductivity of soils and rocks. As all rocks are more or less porous and as the pores are filled with liquids and gases, their electrical conductivity depends on the following factors:

1. Electrical conductivity of minerals composing the rock.

2. Electrical conductivity of the liquid filling the pores.

3. Proportion of the volume of liquid filling pores of the rocks to the solid rock material.

4. Shape of the pores.

5. Arrangement of the particles of gas and liquids.

6. Temperatures.

The present investigation deals with rocks which have no conducting minerals; therefore, factor 1 is not important.

The liquids contained in soils and rocks in most cases are waters but sometimes are oil. As oil is a nonconductor of electricity, the impregnating waters will determine the electrical conductivity of soils and rocks.

Accordingly the author divides the discussion presented in this paper into the following headings:

1. Occurrence and composition of waters in soils and rocks.

2. Water content of soils and rocks.

3. Determination of electrical conductivity of soils and rocks.

4. Order of magnitude of conductivity of soils and rocks.

Examples of direct determination of the electrical conductivity of rocks by different methods are given. However, as it is generally difficult, or impossible, to determine the electrical conductivity directly the author developed an indirect method to study the conductivity of rocks. The principles of this indirect method are explained.

Tables, diagrams, and 44 references complete the article.—W. Ayvazoglou.

(668) ELECTRICAL CORING; A METHOD OF DETERMINING BOTTOM-HOLE DATA BY ELECTRICAL MEASUREMENTS

By C. and M. Schlumberger and E. G. Leonardon

American Institute Mining and Metallurgical Engineers, New York, Technical Publication 462, 1932, 38 pp.

The authors have described a series of methods and apparatus which permit tho study of the formations penetrated by drill holes. They gave to the ensemble of these processes the name of "electrical coring" from its analogy with mechanical coring, which has been thus far the only tool at the disposal of the geologist and production engineer to secure subsurface data in drilling exploration.

The process comprises a series of measurements carried out in situ by lowering the measuring apparatus into the drill hole. Thus, numerous bottom-hole data are obtained, namely: The electrical resistivity, the porosity, the temperature of the rocks, the direction of the dip of the formations, the resistivity of the muds, and the inclination and direction of the hole.

In the examination and identification of formations, the use of two physical parameters has been introduced, the electrical resistivity and the porosity. The two parameters, measured inside the hole, make it possible to obtain numerous and reliable stratigraphical conditions. These correlations advantageously replace, in most cases, those obtained with the expensive and slow procedure of mechanical coring.

Strata saturated with oil or natural gas show up very distinctly on the resistivity diagrams on account of their great electrical resistivity and on the porosity diagrams on account of their great permeability. Their identification, therefore, is particularly clear. Furthermore, the numerical value of the resistivity, measured in a given oil-bearing stratum, generally gives a satisfactory idea of its future production. Finally, it is possible, with the use of a simple procedure, to determine the rock pressure inside a porous stratum.--Authors' abstract.

(669) RESULTS OF EARTH RESISTIVITY SURVEY ON VARIOUS GEOLOGIC STRUCTURES IN ILLINOIS

By M. King Hubbert

American Institute Mining and Metallurgical Engineers, New York, Technical Publication 463, 1932, 23 pp.

Three types of problems were investigated by the earth-resistivity method of Wenner and that of Gish and Rooney: (1) Faults in Paleozoic sediments, (2) gravel deposits and ground-water supply in glacial drift, and (3) anticlines buried beneath glacial drift.

It was found that faults could be discovered, especially if the outcrops on the opposite sides were different or if a considerable shear zone was associated with the fault plane. Some minor faults shown on the geological map that had the same formation outcropping on both sides were not detected.

It was found that major gravel deposits having fairly sharp boundaries, even if full of water and buried under considerable till cover, could be detected readily because of their relatively higher specific resistivity. It was found also that the one buried anticline tested gave a pronounced anomaly, which coincided with the anticlinal axis as independently determined from well-log data. Attempts at precise depth determinations have not as yet been successful.

It seems, therefore, that for the types of problems investigated this method has, in spite of its limitations, demonstrated itself to be a highly useful and economical means of at least doing reconnaissance work and is capable of supplanting, to a large extent, more expensive means of investigation.--Author's abstract.

(670) LOCATION AND STUDY OF PIPE LINE CORROSION BY SURFACE ELECTRICAL MEASUREMENTS

By C. and M. Schlumberger and E. G. Leonardon

American Institute Mining and Metallurgical Engineers, New York, Technical Publication 476, 1932, 24 pp.

In the past decade numerous scientists and industrial organizations have undertaken extensive investigations of the corrosion of pipe lines. A first

result of this work was to bring into evidence the principal laws that seem to condition the chemical corrosion of buried metallic masses, particularly the relations which exist between the corrosion, the corrosiveness of the soil, and its resistivity.

At the same time numerous surveys of pipe lines were performed. They showed that the zones of corrosion are intimately connected with the existence of long-line electric currents flowing through the pipes. In the investigations thus far carried out on the subject, the operators have made their measurements by making direct contacts with the pipe or by effecting the measurements in its immediate vicinity. All of this work required, as a rule, expensive and lengthy excavations.

The authors, taking advantage of the experience acquired during numerous years of electrical prospecting, have approached the problem of corrosion from a new angle and evolved a method and apparatus which permit the survey of pipe lines by electrical measurements carried out at the surface and which do not necessitate any excavating. The zones of corrosion, or "hot spots" of the pipes, can be located under all circumstances, including the often very complicated case offered by the urban pipe-line systems where strong stray currents varying rapidly in time are prevalent.

The apparatus is neither heavy nor cumbersome. It can be operated easily by one or two observers and one helper. Since the measurements are made entirely at the surface, the method is rapid and inexpensive. Already, several extensive investigations of practical and industrial problems have been carried out with satisfactory results.--Authors' abstract.

(671) INTERPRETATION OF RESISTIVITY MEASUREMENTS

By G. F. Tagg

American Institute Mining and Metallurgical Engineers, New York, Technical Publication 477, 1932, 13 pp.

This paper deals with a method of interpreting resistivity measurements for the simple case of a single horizontal stratum. The theory of the method is discussed and an experimental survey, carried out by the author to test out this theory, is described. A suitable site where there was a single underlying stratum practically horizontal and where the surface was approximately level was chosen; it was on Cleeve Hill Common near Cheltenham, Gloucestershire. Tests were carried out at two stations. The apparatus consisted of a megger ground tester and a potentiometer-milliammeter equipment with reversing commutators.

Results obtained at both stations are given in diagrams and tables.

The author concludes: "From the experimental results it appears that the method of interpretation based on theoretical considerations given yields satisfactory results when the geological conditions conform to those assumed in the mathematical investigation. It is possible that similar theories and methods of interpretation can be worked out for other forms of geological structure."--W. Ayvazoglou.

5. RADIOACTIVE METHODS

(672) EINE NEUE METHODE ZUR BESTIMMUNG DES EMANATIONSGEHALTES DER ATMOSPHÄRE

(A NEW METHOD FOR THE DETERMINATION OF THE EMANATION CONTENT OF THE ATMOSPHERE)

By W. Messerschmidt

Physikalische Zeitschrift, Leipzig, vol. 32, No. 14, 1931, pp. 548-549.

The method is based on the direct measurement of conductivity of the outside air compressed in the ionization chamber. To measure the emanation content of the atmosphere the outside air is drawn into the ionization chamber by means of a copper tube from a height of 20 meters and compressed to 20 atmospheres. The ionization chamber has a volume of 8 liters and is protected by a 12-centimeter thick lead cover. The ionization current consists of α ionizations of RaEm, RaA and RaC'.

The average accuracy of measurement is about 4 per cent. The results of some tests are discussed. A schematic design of the apparatus is given.--W. Ayvazoglou.

(673) GEIGER-MÜLLER TUBE AS A QUANTITATIVE ION-COUNTER

By J. A. Van den Akker

Review of Scientific Instruments, vol. 1, November, 1930, pp. 672-683.

It was found that the new Geiger-Müller tube is a quantitative detector of individual ions if certain precautions are observed. A bare polished wire along the axis of the tube gives satisfactory results if the resistance and capacity used in the arrangement are given suitable values. The Geiger-Müller tube is quantitative only where the rate of counting is moderate and is particularly accurate where the rate of counting is very low.

The spurious discharges that enter at higher rates of counting seem to be part of the intrinsic nature of the Geiger-Müller tube in its present form; even for large tubes the rate of counting can perhaps never exceed about 300 per minute.

The spurious discharges may be caused by the momentary rise of the field gradient to abnormally high values due to temporary disruptions of the ion layers in a tube.--J. J. Stewart; reprinted from Science Abstracts, Section A, vol. 34, No. 401, 1931.

6. GEOTHERMAL METHODS

(674) REPORT ON FOUR YEARS' STUDY OF EARTH TEMPERATURES IN WELLS

By Charles E. Kern

The Oil and Gas Journal, Tulsa, vol. 29, No. 33, 1931, pp. 54 and 104.

This report, made to the American Petroleum Institute, covers four years of work to indicate a possible method for assisting in the location of oil domes.

The recording of earth temperatures undertaken with the financial support of the American Petroleum Institute has been under the general supervision of Van Orstrand, who initiated, about 10 years ago, the geothermal method for locating oil domes.

Temperature tests taken in approximately 350 oil wells in Texas, Oklahoma, California, and New Mexico have served as foundations for drawing maps and diagrams. They show the relation between temperature and structure in many instances.

The difficulties in interpreting thermal readings are discussed.--W. Ayvazoglou.

7. UNCLASSIFIED METHODS

(675) GEOPHYSICAL SURVEYS OF THE HULL-GLOUCESTER AND HAZELDEAN FAULTS

By A. H. Miller, C. A. French and M. E. Wilson

Canada Department of Mines, Geological Survey, Memoir 165, 1931, pp. 190-225.

In this article the application of the torsion balance to Canadian problems and conditions is discussed in connection with the general problem of methods of geophysical prospecting. The contents are as follows:

Chapter I. Introduction.

Chapter II. Geology of the Ottawa District, by M. E. Wilson.

Chapter III. Torsion balance survey of the Hull-Gloucester and Hazeldean faults (the gradient and the horizontal directing tendency in the case of a fault; results at Leitrum and Hazeldean; and behavior of the instrument), by A. H. Muller.

Chapter IV. Magnetic surveys of the Hull-Gloucester and Hazeldean faults (geological structure; theoretical results; instruments; magnetic results; results at Hazeldean, and local disturbance forces), by C. A. French.

In the conclusion to this chapter French says that there was no positive evidence of a fault at the points indicated by the torsion balance, although there were indications of a discontinuity in the vicinity of the fault and probably associated in some way with it. At Hazeldean, on the other hand, there were definite indications of the existence of a fault. Considering the fact that the interpretation of magnetic anomalies is a difficult question it is doubtful is a more definite response could be expected.--W. Ayvazoglou.

(676) THE GEOPHYSICAL DISCUSSIONS

By Sherwin F. Kelly

Mining and Metallurgy, New York, vol. 12, No. 291, 1931, pp. 151-153.

The article contains brief reports on geophysical papers presented at two sessions held by the Committee of the American Institute of Mining and Metallurgical Engineers.

The following papers were discussed:

1. Resistivity measurements upon artificial beds, by J. H. Swartz (Geophys. Abs. 23).

2. Experimental observations of electromagnetic absorption at the Mammoth Cave of Kentucky, by J. W. Joyce (Geophys. Abs. 25).

3. A new development in electrical prospecting, by H. Lundberg and Th. Zuschlag (Geophys. Abs. 25).

4. Electrical exploration applied to geological problems in civil engineering, by E. Leonardon (Geophys. Abs. 24).

5. Mathematical theory of electrical flow in stratified media, by D. Ehrenburg and R. Watson (Geophys. Abs. 24).

6. Analysis of some seismic prospecting field data, by L. Don Leet and M. Ewing. The authors presented some formulas for time-distance curves and emphasized the importance of curved paths of seismic impulses.

7. A magnetic method of estimating the height of some buried magnetic bodies, by A. S. Eve (Geophys. Abs. 24).

8. Discrimination in applying geophysics, by Sherwin F. Kelly (Geoph. Abs. 30).

9. Practical geomagnetic exploration with the Hotchkiss superdip, by Noel H. Stearn (Geophys. Abs. 19).

10. Method for determining the magnetic susceptibility of core samples, by William M. Barret (Geophys. Abs. 24).

11. Geophysical examination of Meteor Crater, Ariz., by J. J. Jakosky, C. H. Wilson and J. W. Daly (Geoph. Abs. 25).

In closing the meeting, McLaughlin, president, voiced a strong request for more papers dealing with the intricacies of interpretating the field readings and with practical illustrations from experience of how conclusions are drawn from the data obtained.--W. Ayvazoglou.

(677) SCIENCE AND PRINCIPLES OF GEOPHYSICS

By C. A. Heiland

Engineering and Mining Journal, New York, vol. 133, No. 1, 1932, p. 48.

In this article Heiland gives three reviews on the following volumes of Dr. B. Gutenberg's Handbuch der Geophysik:

Vol. 3, pt. 1, 570 pp. Berlin: Gebrüder Borntrager; R.M. 48.00. This volume is chiefly geological.

Vol. 4, pt. 1, 298 pp. Theory of earthquake waves; observations; ground unrests. Berlin: Gebrüder Borntrager; R.M. 22.00.

Vol. 4, pt. 2, 286 pp. Dr. H. P. Berlage, jr., Seismometer, evaluation of diagrams; Prof. Dr. A. Sieberg, Geology of earthquakes. Berlin: Gebrüder Borntrager; R.M. 30.00.--W. Ayvazoglou.

(678) LEAST SQUARES IN PRACTICAL GEOPHYSICS

By Irwin Roman

Transactions American Institute Mining and Metallurgical Engineers, Geophysical prospecting, 1932, pp. 460-506.

The method of least squares has been successfully used in practical geophysics to improve the interpretation of the observed data by reducing the effects of inaccuracies in the measurements. This paper discusses problems selected from (1) sound surveying, (2) seismograph reflection exploration, (3) torsion balance surveying.

For each case the general problem is formulated, and the results are applied to numerical data to illustrate the actual forms of calculation.--Author's abstract.

(679) FINDING MINERALS BY PHYSICAL METHODS

By Captain H. Shaw

Discovery, London, vol. 12, No. 136, 1931, pp. 120-124.

The author describes the progress in geophysical methods of prospecting for mineral deposits.

A brief historical outline from the early divining rods, described for the first time in Agricola's "De Re Matallica" (1556), translated by H. Hoover, is given.

The relation of the physical properties of rocks to the geophysical methods of prospecting is mentioned, and apparatus used is discussed.--W. Ayvazoglou.

(680) LES MÉTHODES GÉOPHYSIQUES DE PROSPECTION DU SOUS-SOL

(GEOPHYSICAL METHODS FOR PROSPECTING THE SUBSOIL)

By A. Vigneron

La Nature, Paris, No. 2867, October, 1931, pp. 341-349.

The author gives a description of the principles of geophysical prospecting. Gravimetrical, magnetic, electrical, and seismic methods are discussed and explained by diagrams and photographic pictures of apparatus. Geothermal and radioactive methods are mentioned briefly.

In conclusion the author says that only the gravimetrical, magnetic, and electrical methods have a general application; the seismic method may be capable of giving useful indications but needs to be improved.--W. Ayvazoglou.

(681) APPLICATION OF GEOPHYSICAL METHODS OF PROSPECTING IN THE OIL FIELDS IN AMERICA (IN RUSSIAN)

By B. V. Numerov

Neftianoe Khoziaistvo, Moscow, vol. 21, No. 8-9, 1931, pp. 96-104.

In this article Numerov gives a detailed report on geophysical work carried out in the United States by oil companies. The report is based on information obtained by Numerov during his visit here in 1929. The following oil-bearing regions are considered: Pennsylvania, Oklahoma and Arkansas, East Texas, West Texas, the region to the east of the Rocky Mountains (south Texas and Louisiana), and California.

According to Numerov's information the number of instruments used for various geophysical work by the oil companies is as follows: 200 magnetometers, 150 variometers, and 175 seismographs; also, there are 5 electrical parties and only 1 or 2 pendulum apparatus.

The great success in discovering salt domes is noticed. The increasing application of paleontology for studying the geological structure of the earth's crust, especially by oil companies in California, is mentioned.

The following scheme of conducting geophysical work in the U.S.S.R., based on the experience obtained in the United States, is proposed:

1. Drawing up of maps of oil-bearing regions based on the existing geological maps and on the analogous locations of deposits discovered in the United States and other countries.

2. Aerial survey of single regions on a scale of 1:20,000 and general maps on a scale of 1:100,000, based on astronomic points 50 kilometers apart.

3. General gravitational and magnetic survey.

4. Geological observations and utilization of the existing wells near the anomalous regions.

5. Detailed geophysical survey, gravitational, seismic, and, to a less extent, electrical.

6. Detailed level survey and test drilling in fixed regions.

7. Deep drilling based on geological data and on the resulgs of geophysical prospecting.--W. Ayvazoglou.

(682) CHOICE OF GEOPHYSICAL METHODS IN PROSPECTING FOR OIL REPORTS

By E. DeGolyer

Transactions American Institute Mining and Metallurgical Engineers, Geophysical Prospecting, New York, 1932, pp. 9-24.

After a brief discussion on geophysical methods in oil finding (magnetic, gravity, electrical, and seismic) the author proceeds to a consideration of the choice of methods. According to his statement there are four major factors to be considered:

1. Usableness of the method. Can it be expected to give its own type of data in the area under consideration?

2. Cost: (a) On a unit-area basis and (b) on a time-operating unit basis.

3. Time-speed of coverage of area.

4. Value of results expected.

An approximate tabulation of costs is given in the following table:

Costs of geophysical surveys

	Crew per month	Costs per station	Costs per acre
Magnetic	$1,000	$ 1.30 - $ 1.50	$0.0028
Torsion balance ..	4,500	24.00 - 25.00	.0450
Pendulum	5,000	30.00 - 40.00	.0720
Electric	6,000	25.00 - 30.00	.0550
Seismic-refraction	15,000		.0800
Seismic-reflection	12,000	80.00 - 100.00	.1800

In the writer's opinion and experience, and in the present state of development of the geophysical methods for areas where the occurrence of oil pools is controlled by normal folding--anticlines, faulted structures, etc.,--the results obtained from seismic surveys, reflection method, are more definite and of greater value than any other type of geophysical information obtainable.

The writer's second choice for all areas and first choice for areas in which the seismic methods are not usable is the gravimetric method, generally a torsion-balance survey.

The performance of the electrical methods in the oil industry has not been impressive.

The results of magnetic surveys are not susceptible of interpretations sufficiently definite in terms of geologic structure to give them any considerable value to the oil prospector, except in few very special cases.

In conclusion the author stresses the point that the fundamental and most important part of all geophysical work is the proper interpretation in geologic terms of the physical data secured from field observations, that is, the cooperation between the physicist and the geologist.--W. Ayvazoglou.

(683) MODERN METHODS OF GEOPHYSICAL PROSPECTING

Editorial note

The Rhodesian Mining Journal, Johannesburg, vol. 5, No. 52, 1931, pp. 491-495.

After a brief account of geophysical methods of determining the presence of minerals and the principles on which the practice is based, the author discusses various applications of these methods in South Africa. Electrical and magnetic methods are considered the most suitable ones for South Africa. Favorable conditions for tests and results of geophysical survey at Messina are described. Effects of geologic anomalies and advantages of electrical prospecting are examined. The costs of magnetic surveys, under normal conditions and providing the distances between stations are not too great, are estimated to be from 4 s. to 8 s. per station.--W. Ayvazoglou.

(684) SUMMARIES OF RESULTS FROM GEOPHYSICAL SURVEYS AT VARIOUS PROPERTIES

By Donald H. McLaughlin

Transactions American Institute Mining and Metallurgical Engineers, Geophysical Prospecting, 1932, pp. 24-47.

In the introduction to the article, written by McLaughlin, he mentions the desirability of interchanging opinions concerning results from actual cases. In connection with this, McLaughlin gives a few paragraphs from the following letters: (1) Allen W. Pinger's of the New Jersey Zinc Co. on the application of the methods at a number of its properties. (2) J. W. Weitzenkorn's "Success with molybdenite deposits," a summary of the work done at Questa, N. Mex. (3) "Indications in sericite schists," information from L. G. Morrell concerning a geological and electrical survey in a region of sericite schists impregnated with disseminated sulphides. (4) "Reports from Canada," information furnished by George Bury concerning electrical surveys using a high frequency method on two properties in Canada, the Abbey Mines (Ltd.), and the St. Lawrence Metals (Ltd.); by Joseph Errington, of Toronto, on favorable results in the Sudbury Basin; by C. V. Clausen on the failure to secure definite results at the Treadwell-Yukon properties; and by W. H. Emens of the Mining Corporation of Canada concerning results at various properties of this company. (5) Letters on "Limitations of geophysical work" from Robert E. Tally, general manager of the United Verde Copper Co.; Fred Searls of the Newmont Mining Corporation; and J. M. Snow of the Tintic Standard Mining Co.

In the summaries of results from geophysical surveys at various properties the following articles are also included:

1. Kennecott Mines, Alaska, by Alan M. Bateman (Geophys. Abs. 19).

2. Southern British Columbia, by J. J. O'Neill (Geophys. Abs. 19).

3. Gull Lake, North Central Newfoundland, by E. Y. Dougherty (Geophys. Abs. 19).--W. Ayvazoglou.

(685) THE DETECTION OF MINERALIZED DEPOSITS BY GEOPHYSICAL METHODS

By A. G. Boyden

The Mining and Industrial Magazine of Southern Africa, Johannesburg, vol. 12, No. 4, 1931, pp. 109-110.

Prospecting by means of sound waves, earth tremors, and radioactive properties, is mentioned. Electrical methods are considered to be the most successful ones for locating orebodies. The principles of applying (1) the self-potential, (2) the surface-potential, and (3) the inductive methods are discussed briefly. Results of geophysical survey of Northern Transvaal (Messina) are examined.--W. Ayvazoglou.

(686) POSSIBLE NEW FIELD IN CALCASIEU AREA

By Neil Williams

The Oil and Gas Journal, Tulsa, vol. 29, No. 37, 1931, pp. 46 and 180.

Indications of a possible new field on the Gulf Coast, by gas showings in a test in an Iowa geophysical salt-dome prospect in the Calcasieu and Jefferson Davis Parish line, were given. This Iowa prospect was discovered by seismograph during 1929.

Another new field, although at present not of commercial importance, was opened in Cameron Parish; it was also discovered by seismograph in 1928.

The Mykawa, a torsion-balance salt-dome prospect, is also of interest. It was opened for production during 1929 by a gas well, which later sprayed a little oil.--W. Ayvazoglou.

(687) SECOND WELL AT CLEAR LAKE IMPORTANT

By Neil Williams

The Oil and Gas Journal, Tulsa, vol. 30, No. 7, 1931, p. 40.

The author reports on the opening of three new fields (Lockwood, Clear Lake; Lake Grand Ecuille, southern Plaquemines Parish; and Bayou Choctaw, Iberville Parish) in rapid succession. This indicates, according to Williams, the potential possibilities of new production on the Gulf Coast and at the same time speaks well for geophysical science, to which the discoveries are credited.

Indications of still another new geophysical field on the Iowa prospect, which straddles the Calcasieu-Jefferson Davis Parish line, is mentioned.--W. Ayvazoglou.

(688) COAST OPERATORS WATCHING MANVEL

By Neil Williams

The Oil and Gas Journal, Tulsa, vol. 30, No. 17, 1931, p. 39.

The Manvel prospect, discovered by torsion balance in 1929, is approximately 20 miles south of Houston.

The outcome of the drilling test, which promises the opening of another important new field, is watched with considerable interest.

Another geophysical prospect was definitely proved up as a dome when Pure Oil Co. drilled into rock salt in its No. 1 Alliance Trust Co., Gueydan, Vermillion Parish, Louisiana. Gueydan was discovered by seismograph in 1929. --W. Ayvazoglou.

8. GEOLOGY

(689) DIE SCHWERE AM OSTRAND DES FENNOSKANDISCHEN SCHILDES

(GRAVITY ON THE EASTERN BORDER OF THE FENNOSKANDIAN SHIELD)

By Robert Swinner

Gerlands Beiträge zur Geophysik, Leipzig, vol. 34, No. 3, 1931, pp. 436-472.

The eastern part of the Fennoskandian shield and the adjacent part of the Russian plateau (Finland, northwest Russia, and the Baltic States) show remarkable relations between the geologic structure and the distribution of gravity. Curves of equal gravity anomalies (reduced according to Bouguer), the so-called isogams, were drawn on a geologic map. The result is as follows:

The granites (the Rapakivi massif of Viborg, Aaland, Björneborg, as well as the older granites of Vaasa and Uleaborg) are distinguished by minus gravity, corresponding to the fact that granite, having a smaller specific weight than its surroundings, is a negative disturbing mass. Why the central granite of Finland is an exception is to be investigated more closely.

Moreover, the isogams render very closely the folds that can be distinguished in the structure of the crystalline rocks, especially in the Karelian and southern Finnish mountains; therefore, from the further direction of these isogams a conclusion can be drawn on the continuation of these mountain structures under the Russian plateau. According to this, the Karelian mountains would continue as follows: By one branch towards the Lower Dvina, bordering the "Block of the White Sea"; by another branch, in a sigmoid turn northwards around Lake Ilmen, to the "Wall of Vishny-Volochek." From the Southern Finnish Mountains a connection in the form of a curve leading from the southwestern point of Finland across Esthonia and the Duna district towards the "Scythic Wall" can be supposed.

On the other hand, a connection between the distribution of gravity (represented by the isogams) and the quarternary uplift (represented by the isobases = lines of equal uplift) can scarcely be proved.--Author's abstract translated by W. Ayvazoglou.

(690) EAST HACKBERRY SALT DOME, CAMERON PARISH, LOUISIANA

By A. J. Bauernschmidt

Bulletin of the American Association of Petroleum Geologists, Tulsa, vol. 15, No. 3, 1931, pp. 247-256.

The East Hackberry salt dome, Cameron Parish, Louisiana, was discovered in 1926 by means of the seismograph.

It is a knob on a large salt mass which includes West Hackberry. Most of the oil is produced from Miocene sands that are approximately 3,900 feet in depth, but the producing sands overlying the cap rock are probably Pliocene.

The oil (18° - 24° Bé gravity) occurs in a narrow zone on the south flank of the dome and in a small area overlying the cap. The initial production of the wells ranges from 25 to 3,000 barrels of oil per day, and from 1927 to the end of 1930 a little more than 2,000,000 barrels of oil has been produced from 35 wells.--Author's abstract.

(691) THE GEOLOGY AND ORE DEPOSITS OF BUCHANS, NEWFOUNDLAND

By W. H. Newhouse

Economic Geology, Lancaster, Pa., vol. 26, No. 4, 1931, pp. 399-414.

A geological description of ore deposits of Buchans found by geophysical prospecting is given.

The history of the discovery of the orebodies has been discussed in the following papers:

1. Lundberg, H., "Recent results in electrical prospecting for ore." Volume on Geophysical Prospecting, Am. Inst. Min. and Met. Eng., pp. 118-122.

2. Snelgrove, A. H., "The geology of the central mineral belt of Newfoundland." Canadian Min. and Met. Bull. 197, 1928, pp. 1057-1127. --W. Ayvazoglou.

9. NEW BOOKS

(692) American Institute of Mining and Metallurgical Engineers, Transactions. Geophysical Prospecting, 1932. Published by the Institute at the office of the secretary, 29 West 39th Street, New York, N. Y. This volume is the second of a series devoted to papers and discussions on geophysical prospecting. The two volumes are as follows: 1929, Geophysical Prospecting; 1932, Trans. Am. Inst. Min. and Met. Eng., Geophysical Prospecting. This volume contains papers and discussions presented at the New York meetings in February, 1929, 1930, 1931, and 1932. Contents:
General: Choice of geophysical methods in prospecting for oil deposits, by E. DeGolyer (Geophys. Abs. 35); Summaries of results from geophysical surveys at various properties (Geophys. Abs. 35).
Resistivity methods: A new development in electrical prospecting, by Hans Lundberg and Theodor Zuschlag (Geophys. Abs. 25); Geophysical examination of Meteor Crater, Ariz., by J. J. Jakosky, C. H. Wilson, and J. W. Daly (Geophys. Abs. 25); Electrical exploration applied to geological problems in civil engineering, by E. G. Leonardon (Geophys. Abs. 24); Applying the megger ground tester in electrical exploration, by Bela Low, Sherwin F. Kelly, and William B. Creagmile (Geophys. Abs. 32); Depth of

investigation attainable by potential methods of electrical exploration, by C. and M. Schlumberger (Geophys. Abs. 12); Electrical studies of the earth's crust at great depths, by C. and M. Schlumberger (Geophys. Abs. 12); A uniform expression for resistivity, by Sherwin F. Kelly (Geophys. Abs. 35).
Electromagnetic methods: Mapping oil structures by the Sundberg method, by Theodor Zuschlag (Geophys. Abs. 12); Absorption of electromagnetic induction and radiation by rocks, by A. S. Eve (Geophys. Abs. 12).
Magnetic methods: Practical geomagnetic exploration with the Hotchkiss Superdip, by Noel H. Stearn; A magnetic method of estimating the height of some buried magnetic bodies, by A. S. Eve (Geophys. Abs. 24); A method for determining magnetic susceptibility of core samples, by William M. Barret (Geophys. Abs. 24).
Seismic methods: A new geophone, by C. A. Heiland (Geophys. Abs. 17); Seismic propagation paths, by Maurice Ewing and L. Don Leet (Geophys. Abs. 10); Comparison of two methods for interpretation of seismic time-distance graphs which are smooth curves, by Maurice Ewing and L. Don Leet (Geophys. Abs. 35).
Gravitational methods: Interpretation of gravitational anomalies, I, by H. Shaw (Geophys. Abs. 5); Interpretation of gravitational anomalies, II, by H. Shaw (Geophys. Abs. 17).
Theoretical studies: Effect of impregnating waters on electrical conductivity of soils and rocks, by Karl Sundberg (Geophys. Abs. 35); A theoretical study of apparent resistivity in surface potential methods, by J. N. Hummel (Geophys. Abs. 25); Mathematical theory of electrical flow in stratified media with horizontal, homogeneous, and isotropic layers, by D. O. Ehrenburg and R. J. Watson (Geophys. Abs. 24); Observed and theoretical electromagnetic model response of conducting spheres, by L. B. Slichter (Geophys. Abs. 15); Least square in practical geophysics, by Irwin Roman (Geophys. Abs. 35).

(693) Maurain, Ch. Annales de l'Institut de Physique de Globe de l'Université de Paris et du Bureau Central de Magnétisme Torrestre, vol. 9, 1931, 206 pp. Paris, Presses Universitaires de France, 49, Boulevard St.-Michel. Contents of the book: (1) Magnetic observations made in Val-Joyeux during 1929, by L. Éblé; (2) Magnetic observations made at the observatory in Nantes during 1929, by E. Tabesse; (3) Anomalies of the terrestrial magnetic field in France, by E. Mathias, Ch. Maurain, L. Éblé and Miss G. Homery; (4) Magnetic measurements in the Hautes-Pyrénées, le Gers et la Haute Garonne, by E. Mathias; (5) Magnetic measurements in the Haute-Marne, the Cote-d'Or and the Aube, by E. Mathias; (6) Magnetic measurements in the Allier and the Puy-de-Dome, by E. Mathias; (7) Magnetic measurements in the Creuse, the Dordogne, and the Haute-Vienne, by E. Mathias; (8) Secular variations of the distribution of magnetic elements; application for tracing maps, by L. Éblé; (9) Study on the variation of the declination in France since 1681, by Miss G. Homery; (10) Remarks on the combination of the difference, by H. Labrouste; (11) On the periodic displacements of the stationary mass of Wiechert's seismograph, by L. Génaux and R. Guilhen; (12) Actinometric studies according to the documents obtained from the Parc Saint-Maur Observatory, by C. E. Brazier;

(13) Actinometric observations made at the Parc Saint-Maur Observatory in 1929, by C. E. Brazier; (14) Summary of meteorological observations made at the Parc Saint-Maur Observatory in 1929, by C. E. Brazier; (15) Summary of seismic observations made at the Parc Saint-Maur Observatory in 1929, by C. E. Brazier and L. Génaux; (16) Electric charge sustained by the air, by E. Salles; (17) Magnetic observations made at the Observatory of Tananarive (Madagascar) during 1929, by Poisson and Delpeut; (18) On the superficial transversal waves (Love-waves), by J. Coulomb; (19) Measurements of electric conductivity in the atmosphere of Paris, by Mrs. F. Bayard-Duclaux; (20) Magnetic observations in Syria. Observations on electric field and on the deprivation in Syria, by J. Chevrier; (21) Principal magnetic perturbations in 1929 (graphics obtained from register lists in Val-Joyeux).

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I. C. 6607

April, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING, TREATMENT METHODS AND COSTS AT THE PLANT OF THE CONSOLIDATED ROCK PRODUCTS CO., DURBIN, CALIF.

BY

Harry D. Jumper

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING, TREATMENT METHODS, AND COSTS AT THE PLANT OF THE CONSOLIDATED ROCK PRODUCTS CO., DURBIN, CALIF.[1]

By Harry D. Jumper[2]

INTRODUCTION

This paper describing the methods of mining a sand, gravel, and bowlder deposit 20 miles east of Los Angeles, and the classification and preparation of these materials for market, is one of a series being prepared for and published by the United States Bureau of Mines on methods and cost of mining sand and gravel. These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total costs. It is recognized that publication of total costs might in many instances cause embarrassment to individual operators as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of the methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue. The description contained herein is typical of this industry in Southern California.

ACKNOWLEDGMENT

The author acknowledges the assistance of L. L. Rogers, vice-president and manager of production and distribution for the Consolidated Rock Products Co., in the preparation of this paper.

HISTORY

The plant to be described was built by the Los Angeles Rock and Gravel Co. in 1923 and was operated by that company until 1925. In the same year all assets of the Los Angeles Co. were purchased by the Union Rock Co., which in 1929 merged with several other companies to form the Consolidated Rock Products Co. The property consists of 500 acres of gravel, a screening and crushing plant, 3 miles of standard-gage railroad connecting the plant yards

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6607."

2 One of the consulting engineers, U. S. Bureau of Mines, and engineer, Consolidated Rock Products Co.

with the Southern Pacific Railroad, and the distribution bunkers for trucks at that point. One mile of standard-gage track connects the plant yards with the Pacific Electric Railroad. The company owns 5 miles of storage tracks in the yards and all necessary locomotives for the handling of railroad cars. Beside the two rail connections mentioned above, the property is served by excellent paved highways paralleling its north and south sides, which connect with all roads in Southern California. The San Gabriel River which bounds the property on the west is a dry wash nine months of the year, but at times during the rainy season the water is difficult to control and keep from doing damage.

GEOLOGY

The deposit is of alluvial origin and was probably formed ages ago by the San Gabriel River during torrential flood seasons. The deposit is fan shaped, widening from ½ mile at the mouth of the canyon in the San Gabriel Mountains to a semicircle with approximately 6-mile radii. Near the plant the material ranges in size from sand to 12 inches in diameter, but closer to the mountains the maximum size is much larger. There are no distinct layers of gravel or sand to be observed on the exposed faces of the pits or in test holes, or shown by the log of wells drilled in this section. All of these show uniform distribution of sizes. The overburden is negligible, consisting of a 6 to 12 inch layer of light sandy soil which is easily washed away in the plant. The water level is about 110 feet below the surface.

PHYSICAL CHARACTERISTICS

Mining operations have been carried on in an open pit with steam shovels working on different levels. This method will probably be continued until a level slightly above the water is reached, then possibly some hydraulic method will be used. From the logs of wells drilled on this property, which provide the deepest tests we have, the same class of material is proved to a depth of 300 feet, and we have every reason to believe it continues much deeper. The raw material from the pit weighs approximately 118 pounds per cubic foot and its screen analysis is as follows:

	Screen opening	Per cent through screen
Sand	3/8-inch circle	44
Gravel	3-inch circle	33
Average[1]	12-inch circle	23
		100

1 This size is crushed and kept separate from the gravel for sale as crushed rock.

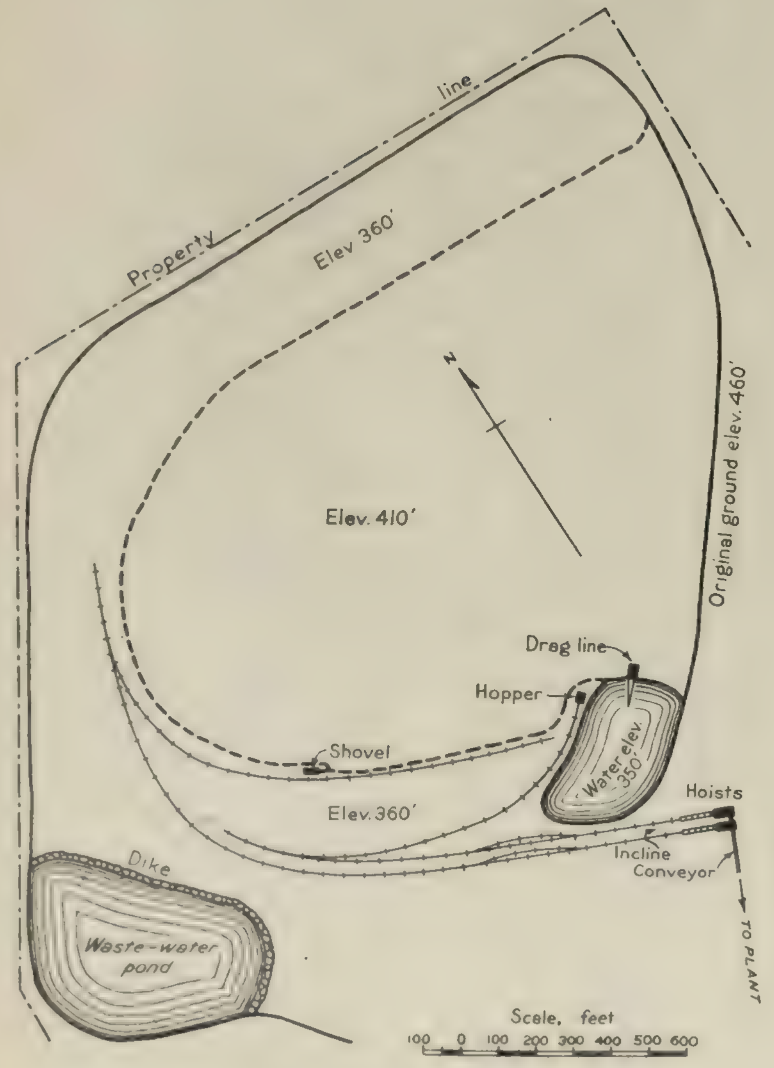

Figure 1.- General plan of pit

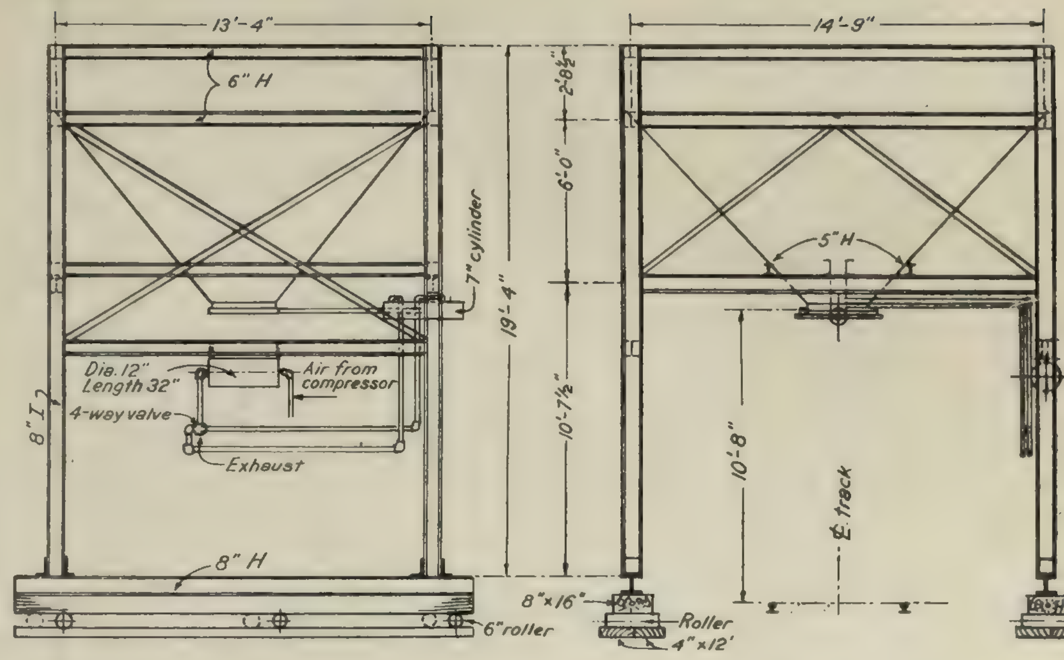

Figure 2.- Cross and longitudinal sections of drag line hopper

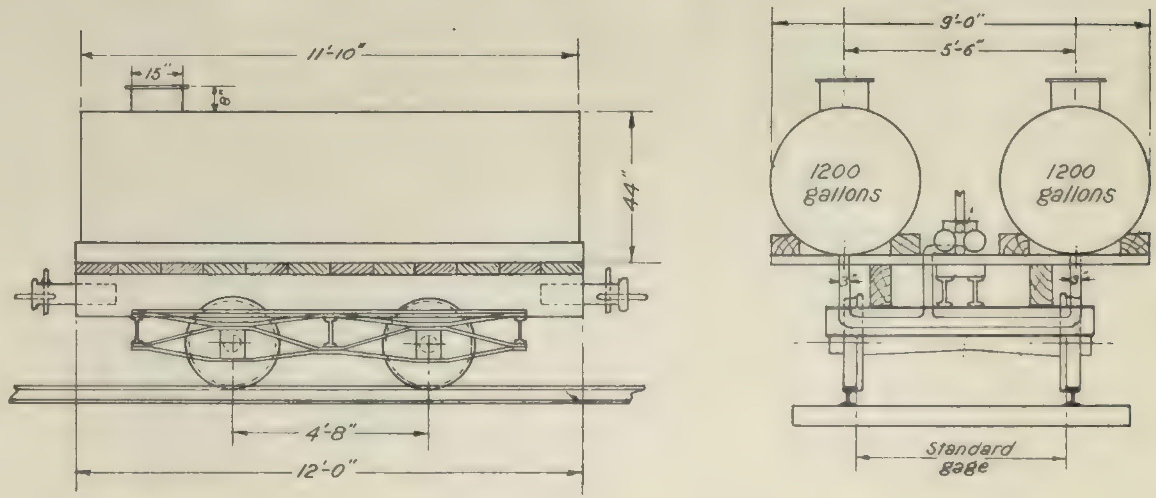

Figure 3.- Side and end view of oil car

Average pit material contains about 69 per cent of silica. The rock fragments are of granitic base and have an average compressive strength of 16,000 pounds per square inch. The DeVal abrasive loss is 2.84, and the French coefficient of wear (ASTM D 2-08) is 14. The specific gravity of the rock is 2.65 and the abrasive loss by dry shot rattler (ASTM D 289-28T, grading B) is 13.6 per cent.

METHODS OF PROSPECTING

Little prospecting was necessary in locating the company's plant, as the deposit had been thoroughly prospected by other companies in the district. However, two 4 by 4 foot pits were dug by hand to water level at a cost of $4 per foot, before actual drilling of wells for the water supply commenced. A record was kept of the material excavated in these pits, and the information obtained from the screen analysis of this material served well in designing the plant to handle the indicated tonnages of the various sizes. The method used in sampling the excavated material was as follows: All the material from each 4 feet of depth, this being the height of each timber set, was piled separately and samples of approximately 500 pounds were taken from each pile and screened through 3/8-inch round holes to determine the sand. The oversize was then put over a screen with 3-inch round perforations, the undersize representing gravel and the oversize material to be crushed. The gravel sample was quartered and then screened to the commercial sizes required by the market, thus giving the percentages of these sizes to be expected.

MINING METHODS

It will be noticed from Figure 1 that the lower bench of the pit is in the shape of a crescent, and when originally opened up, the first shovel cut was made along this line. As operations continued the pit was widened, each cut getting nearer the incline, until the track became so curved that it was impractical to operate over it. Switchbacks were therefore put in to work out this portion of the pit and are in use at present. However, the bottom of the cut where the shovel is working, is considerably lower than the original bench to the north, and as soon as the new cut is of sufficient width to lay out a curved track, this will be done.

The switches on the switchbacks are all of the spring type, so that little time is lost in operating them.

At present the track grades are somewhat irregular, but it is planned to extend the present inclines to a depth which will give a $\frac{1}{2}$-per cent grade from the shovels to the incline. This grade is approximately the general gradient of the property in the same direction, and hence can be easily maintained by keeping a uniform height of pit face.

Loading

Excavating and loading is done by a No. 60 railroad-type steam shovel equipped with a 2-yard dipper and a No. 24 revolving, steam drag line using a 4-yard bucket. This is a large machine weighing approximately 150 tons and is carried on four sets of standard-gage railroad trucks and is moved on double tracks which are made up in short sections. The boom is 110 feet long, and is carried at a 30° angle. In this position the machine has a digging width of 230 feet. The reason for using this drag line in the pit is to provide a place for the disposal of silt and the surplus sand, which can be run by gravity from the plant by pipe line to the pit. At present it requires two 8-inch sand pumps to discharge this waste to the sand dumps. The drag line discharges its bucket into a steel hopper with a capacity of 80 tons or two pit-car loads. The discharge opening of this hopper is 24 inches square and the gate on the bottom is a flat sliding door connected by a rod to the piston of a pneumatic ram 36 inches long by 7 inches in diameter. The gate has a travel of 30 inches and is operated from the ground by a 4-way valve. The necessary air is furnished by a small portable compressor, such as is used in garages or filling stations, driven by a 5-hp. belted motor. A small air container is mounted on this unit for storage and pressure. Figure 2 shows details of the hopper.

TRANSPORTATION

The average haul from the drag-line hopper to the foot of the incline is about $\frac{1}{4}$ mile. A 15-ton saddle-tank steam dinky with four drivers handles three trains of two 40-ton bottom-dump cars or 240 tons per hour. The steam shovel loads directly into cars identical with those described above and these are handled by a 30-ton saddle-tank steam dinky with four drivers, pulling four cars to the train and delivering to the foot of the incline at the rate of eight cars, or 320 tons, per hour. Thus the plant receives a total of 560 tons of raw material every hour. The average haul from the steam shovel is approximately $\frac{3}{4}$ mile.

The track work, consisting of extending, shifting, and maintaining the track in the pit, is done by a small crew which is also used around the plant for general upkeep. All steam equipment uses oil for fuel which is delivered to the plant reservoir in tank cars and is unloaded by gravity. The reservoir is a circular concrete tank with a capacity of four tank-car loads. The top of the tank is 3 feet below the rails. Adjoining and at the bottom of the concrete tank is a 3-inch geared pump used to force the oil through a system of pipe lines to a tank in the pit, and for re-fueling the steam equipment around the plant. Refueling of all plant equipment is done at night by a hostler. The fuel-oil tank in the pit is located near the foot of the inclines on the permanent track. An especially designed tank car is used to carry oil from it to the oil-burning equipment. The details of this tank car are shown in Figure 3. The car consists of two horizontal tanks mounted side by side on the running gear of a standard-gage pit car. Each tank has a dome

through which the tank is filled. The oil is drawn from the bottom of the tanks through a manifold, as shown, and enters the suction side of a duplex pump. The discharge is fitted with a rubber hose through which the oil is pumped to the shovel or locomotive. The switching of the fuel car is done by the locomotive which serves the shovel and is taken on one of its regular trips and left at the shovel to be unloaded. The car is then returned to the oil spout and refilled, where it is left until needed. The fuel oil used comes from local fields and is 18° plus gravity. It costs about 70 cents per barrel f.o.b. the plant. However, the cost fluctuates with the market.

INCLINE AND HOISTS

The two inclines referred to are approximately 500 feet in length, one half of which is on a fill and the other half of which is supported on timber bents. The track is laid on regular 8 by 8 inch by 8-foot ties, both on the fill and timber trestle, except where the dump bin is located. Here the rail is bolted on the top of 18-inch I beams for a 24-foot span. The pit cars dump between these beams, giving a clear opening for the material to drop through without piling up on the ties. All rail is 65-pound steel and all track is standard gage.

The grade of the inclines is 12.5 per cent and their maximum height at the dump bins is about 22 feet above the ground. The hoist operator is stationed in a control room slightly above the trestle, where he has a clear view of the foot of the incline and of the car at all times. The hoists are located below on the ground.

The south hoist, which handles cars from the shovel, is a single-drum machine built locally. The drum is 60 inches in diameter and the hoisting speed at 23.5 r.p.m. is 370 feet per minute. This hoist is driven by a 200-hp. 480-r.p.m. motor through spur gears and pinions. The wire rope used is 6 strand, 19 wire, regular lay made from modified steel, and is $1\frac{1}{4}$ inch in diameter and 700 feet long. It lasts approximately eight months and hoists 650,000 tons.

The north hoist, which handles the cars from the dragline, is also a single-drum machine. Its drum is 48 inches in diameter, and at 26 r.p.m. it gives a car speed of 326 feet per minute. It is driven by a 150-hp., 720-r.p.m. motor by spur gears and pinions. The wire rope is of the same size and construction as for the other hoist. It lasts approximately 11 months and handles 660,000 tons.

FIELD BELT

From the bins previously mentioned, the material is fed on to a 36-inch belt conveyor, 600 feet between centers and traveling 360 feet per minute, by two 36-inch reciprocating feeders, driven by $7\frac{1}{2}$-hp., 750-r.p.m. motors through a spur gear and pinion reduction. (See fig. 4.)

The present belt is of 10-ply 32-ounce duck with a 3/16-inch rubber top increasing in thickness to 5/16 inch for a 28-inch width in the middle and a 1/8-inch rubber cover on the bottom. This belt has been in service 68 months and has handled approximately 4,000,000 tons and there is still considerable wear left in it. The original cost was slightly over $6 per foot.

The belt is horizontal for the first 300 feet, then it curves into an incline of 12° to the head pulley. It is driven by a 75-hp., 690-r.p.m. motor through a gear reduction.

PRIMARY SCALPING SCREEN

The field belt discharges into a 48-inch by 18-foot scalping screen having a 12-foot section of 3-inch circular openings and a 6-foot section of 4½-inch openings. This screen is the trunnion type, using cast-iron head and tail castings with V-shaped lugs to fit 6 by ¾ inch angle irons which connect the two ends. The material passing the 3-inch circular openings amounts to approximately 430 tons per hour and goes to the washing plant via a 30-inch belt conveyor. The material passing the 4½-inch circular openings goes to a box with the discharge of the primary crusher, and thence via a 24-inch conveyor to a secondary scalping screen. The oversize from the 4½-inch screen amounting to about 60 tons per hour goes to the primary crusher.

The screen sections are made up in 6-foot lengths of ½-inch high-carbon steel plates with ¾-inch bridges rolled in semicircles and fastened together with full-length butt straps. These plates have an average life of four months on the 3½-inch section and eight months on the 4½-inch section. The screen is driven by a 25-hp., 750-r.p.m. motor through reduction spur gears, the screen having a ring gear on the tail casting. The screen makes 15 r.p.m. and is set on a slope of 1½ inches per foot.

PRIMARY CRUSHER

The 4½-inch oversize from the primary scalping screen is dropped into a No. 7½ gyratory crusher set at 2½ inches, and driven directly by a 100-hp., 480-r.p.m. motor. The motor and pinion shafts are fastened together by a flexible coupling with bolts designed to break if the crusher jams on tramp iron or excessive feed. This arrangement has so far worked satisfactorily.

TWENTY-FOUR-INCH BELT CONVEYOR

A 24-inch belt conveyor carries the 3½-inch to 4½-inch material from the primary scalping screen and the discharge from the primary crusher to the secondary scalping screen. It is made of 6-ply, 32-ounce duck with 1/8-inch rubber top and 1/32-inch rubber bottom. The inclination of this conveyor is 15°, the speed 250 feet per minute, and it is carried on timber bents. No gravity take-up is used, the tension being adjusted by screw take-ups on the tail-pulley shaft. The life of a new belt is approximately three years, but

a belt which has been in service in other parts of the plant is often used on this conveyor, as the load is light and the centers are short. The conveyor is driven by a 10-hp., 750-r.p.m. motor through a spur-gear reduction at the head pulley.

SECONDARY SCALPING SCREEN

The secondary scalping screen is identical in construction to the primary screen, except as to length, being 48 inches in diameter but only 14 feet long. It revolves at 15 r.p.m. and is set on a slope of 1½ inches per foot. This screen consists of two 7-foot perforated-plate sections rolled in semicircles and fastened together with butt straps. The plates are high-carbon steel and are ½-inch thick. All perforations are 2-inch circles with ¾-inch bridges. The normal feed is approximately 130 tons per hour, 60 tons of which is material reduced by the primary crusher and is screened out here and chuted to a 24-inch crushed-rock conveyor which also carries the material from the secondary crushers to the main screening plant and bunkers. The screen is driven by a 15-hp., 750-r.p.m. motor the same as the primary unit. The life of screen sections is approximately nine months.

SECONDARY CRUSHERS

The oversize or plus 2-inch material, amounting to approximately 70 tons per hour, is split between a 4-foot cone crusher and a 48-inch vertical-disk crusher. These machines are kept set at 3/8 inch to make the maximum of small sizes. A 36-inch horizontal-disk crusher is also included in this unit but is used only in emergencies. These crushers are set on concrete foundations in a line directly beneath the secondary scalping screen. Their feed openings are about 8 feet above ground, thus giving headroom for the discharge of crushed material on the 24-inch crushed-rock belt conveyor which runs through the crusher foundations and directly under the crushers.

The feed to these crushers is collected in a 4-foot cubical timber box built under the end of the scalping screen and having a timber chute 2 feet wide and 2 feet deep which carries the material to the crushers. These chutes are built of 2 by 12 inch planks and on a slope of 45°. Two by fours 2 feet long are spiked to the bottom at 1-foot intervals. The tops of these pieces are armored with 3/8-inch plate and the sides are also lined with ¼-inch plate to a height of eight inches. These chutes are called "riffle chutes," and the theory is that the smaller sizes of rock will lodge behind the cleats and form a rock lining, thus saving wear on timber or chute iron. This type of chute proves successful where headroom will permit a slope of 45° or more but will clog badly if the slope is less.

By splitting the feed to these crushers, the 4-foot cone receives approximately 40 tons per hour and the 48-inch disk receives 30 tons per hour. However, this entire feed has at times been sent to the 4-foot cone without any trouble.

These crushers are all belt driven. The 4-foot cone has a 100-hp., 750-r.p.m. motor giving a crusher speed of 480 to 490 r.p.m.

The 48-inch vertical disk is driven by a 75-hp., 750-r.p.m. motor, giving a crusher speed of 360 r.p.m. The transmission belts to both this crusher and the cone crushers are 14 inches wide, 6 ply, and rubber covered.

The 36-inch horizontal-disk crusher is belt driven by two 30-hp., 750-r.p.m. motors. The speed of the eccentric pulley is 310 r.p.m. and that of the main pulley 135 r.p.m. The transmission belt on the main pulley is 12 inches wide and that on the eccentric pulley is 14 inches wide. Both are 6 ply and rubber covered.

The wearing parts (the outer and inner-bowl liners which are in the crushing zone of the 4-foot cone) are made of manganese steel and have an average life of 9 months or approximately 85,000 tons. The disks in the vertical-disk crusher have a life of about 12 months or approximately 85,000 tons. The inner and outer disk in the horizontal-disk crusher have about the same average life as those above.

TWENTY-FOUR-INCH CRUSHED-ROCK CONVEYOR

This conveyor, supported by a timber trestle, carries the product from the secondary scalping screen and the secondary crushers, approximately 130 tons per hour, to the main screening and washing plant. The conveyor is 280 feet between centers and is inclined 16°. It is driven by a 50-hp., 750-r.p.m. motor at 200 feet per minute by spur-gear reduction. The belt is made of 6-ply, 32-ounce duck and is 24 inches wide, with a 3/16-inch rubber top and 1/16-inch rubber bottom. It is kept tight by a gravity take-up. The life of this belt is approximately three years or 1,170,000 tons.

SCREENING PLANT

The crushed-rock conveyor just described discharges into a steel box 3 feet square and 3 feet deep, from which steel chutes lead to two 48-inch by 30-foot revolving screens. Approximately 200 gallons of water per minute is introduced into this box and carries the material down the chutes into the screens for further washing and cleaning.

These screens are the trunnion type with open heads, the tail castings being solid, with a ring-gear bolted to them for the drive, and are cast with lugs for 8 by 8 by $\frac{3}{4}$ inch angle irons which connect the head and tail castings and also hold the screen sections in place.

The main screen is made up of three sections. The head section is 12 feet long and is made of 5/16-inch steel plate with 3/8-inch bridges between ½-inch circular openings. The second section is 10 feet long and is made of 3/8-inch steel plate with 3/8-inch bridges between 1-1/8-inch circular openings. The third section is 6 feet long and is made of ½-inch steel plate with 5/8-inch bridges, connecting 2-inch circular openings. All the screen plates are rolled in semicircles and fastened together with full-length butt straps. The outside, or dust jacket, is 11 feet long, 60 inches in diameter, and is made of 1/8-inch plate with ¼-inch bridges between 3/16-inch circular openings. These screens are pinion driven by individual 20-hp., 750-r.p.m. motors at 14 r.p.m.

The sizes of crushed rock produced are as follows:

	Size, inches
No. 2 crushed rock	1 to 2
No. 3 crushed rock	½ to 1
No. 4 crushed rock	1/8 to ½

Of a total of 150 tons per hour, one-half goes to each screen. Approximately 25 tons per hour passes the ½-inch screen to the 3/16-inch dust jacket, and about 10 tons per hour passes through as dust and is carried away by the wash water in a flume and dumped in the sand drags. Approximately 75 tons per hour passes the 1-1/8 inch screen and is retained on the ½-inch screen, and 28 tons per hour passes the 2-inch screen and is retained on the 1-1/8-inch screen. This leaves 2 tons of rejects per hour, mostly in the form of slabs, which are put in a bin and returned to the horizontal-disk crusher for further reduction when the bin is filled at the end of every 10 days.

The life of the dust jacket on the main screen is approximately 12 months; the head, second and end sections last 8, 13, and 20 months, respectively.

All crushed rock is washed by sprays in the screens. About 500 gallons of water per minute is used in this process, or approximately 250 gallons per ton. The different materials after being washed and classified, are run by the riffle chutes, previously described, to their respective bins for loading. These chutes are carried to the bottom of the bins, as this has been found to prevent breakage and segregation to a large extent, especially when the bins are nearly empty. By using this method, the material rolls slowly to the bottom and as the bin fills up, it automatically causes the material to run over the sides of the chute and spread through the bin uniformly.

THIRTY-INCH SAND-AND-GRAVEL CONVEYOR

A belt conveyor carries all material passing the first section of 3-inch holes in the primary scalping screen, to the washing plant. It is 30 inches wide, 260 feet between centers, and is made of 8-ply, 32-ounce duck. It is inclined 18° and has a gravity take-up. The life of this belt is about three

years, during which time it handles approximately 1,000,000 tons of material. The belt travels 300 feet per minute and is driven by a 75-hp., 750-r.p.m. motor through spur gears and pinion reduction. The conveyor is carried on timber bents.

WASHING PLANT

The 30-inch sand and gravel conveyor discharges into a steel box 3 feet wide, 6 feet long, and 4 feet deep. In this box is a small 2-way grizzly made up of steel bars 3/8 x $1\frac{1}{2}$ x 36 inches spaced 5/16 inch apart. The bars are set at 45°, one set sloping toward each of the dual screens and the material from the belt striking the apex and splitting. Approximately 400 gallons of water per minute is introduced at this point to wash as much sand as possible to the sand drags located below, thus relieving the washing screens of this load. The remaining sand and gravel goes to two 48-inch by 30-foot screens identical in construction to those described under "Screening Plant."

The main screen is made up of three sections. The head section is 12 feet long and is made of 5/16-inch steel plate with 3/8-inch bridges between $\frac{3}{4}$-inch circular openings. The second section is 8 feet long and is made of 3/8-inch plate with 3/8-inch bridges between $1\frac{1}{4}$-inch circular openings. The third section is 8 feet long and is made of $\frac{1}{2}$-inch plate with 5/8-inch bridges between $1\frac{3}{4}$-inch circular holes. The outside or sand jacket is 11 feet long and is made of $\frac{1}{4}$-inch steel plate with 5/16-inch bridges between 5/16-inch holes. All the above screen sections are rolled in semicircles and fastened together with butt straps.

Approximately 240 tons of the 430 tons per hour fed to these screens is sand. About 50 per cent of this sand, or 120 tons per hour, is washed through the grizzlies directly to the sand drags. The remaining 120 tons is washed through the outside jacket to the sand drags. The minus $\frac{3}{4}$-inch material passing the first section of the screen and retained on the outside jacket -- approximately 70 tons per hour -- is chuted to two 4 by 8 feet cone screens made of 3/8-inch plate with 3/8-inch bridges between $\frac{1}{2}$-inch circular holes. Approximately 20 tons per hour passes these screens and is chuted to storage bins as No. 4 gravel. Approximately 50 tons per hour of plus $\frac{1}{2}$-inch material is rejected from the end of these cone screens and mixes back into the chute with the material passing through the $1\frac{1}{4}$-inch holes of the second section of the main screens. This material amounts to approximately 30 tons per hour, making a total of 80 tons per hour going to the storage bins as No. 3 gravel.

The undersize through the $1\frac{3}{4}$-inch circular holes of the third section, approximately 20 tons per hour from one screen, goes directly by chute to the No. 2 gravel bin, while the 20 tons per hour from the other screen is mixed in one of the chutes with the minus $1\frac{1}{4}$ plus $\frac{1}{2}$ inch material, thus making No. 23 gravel.

The oversize from the end of the screens, minus 3 inches and plus $1\frac{3}{4}$ inches, amounting to 50 tons per hour, is normally chuted to the storage bins for No. 1 gravel; however, chute gates and storage bins are provided whereby a portion of this size can be mixed with the smaller sizes, making a graded aggregate from 3 inches to $\frac{1}{4}$ inch in size, or No. 123 gravel, which is a graded mixture of Nos. 1, 2 and 3 gravels. This mixing is accomplished by chutes which take all or a portion of the material in them to the chute of another size, discharging into it about one-half down, thus giving the material time to mix before reaching the storage bin. A metal slide gate opens or closes these mixing chutes, the rate of flow depending on the width the gate is opened. Mixtures are made to comply with certain specifications and samples are taken from the chutes during the mixing and tested. From these samples corrections in flow are made if necessary, and the chute gates are marked and made fast at the proper openings.

The life of the screens in the washing plant is as follows: Sand jacket, 5 months; first section, 4 months; second section, 7 months; and third section, 10 months. The life of the cone-screen sections is 14 months.

The two washing screens are driven at the tail end by individual 20-hp., 750-r.p.m. motors. The cone screens are driven by a 15-hp., 750-r.p.m. motor, the two being hooked up in tandem with chain and sprockets, one sprocket being on a short shaft where a pulley is belt driven from the motor pulley. The screen speed is 13 r.p.m.

The various commercial sizes shown below are either direct products from the screens or are made by combining two or more sizes in proper proportions by the system of mixing chutes previously described.

Commercial sizes and nomenclature of products

	Inches
No. 1 gravel	$1\frac{1}{2}$ to 3
No. 2 gravel	1 to $1\frac{1}{2}$
No. 3 gravel	$\frac{1}{4}$ to 1
No. 4 gravel	10 mesh to $\frac{1}{4}$
No. 23 gravel	$\frac{1}{4}$ to $1\frac{1}{2}$
No. 123 gravel	$\frac{1}{4}$ to 3
Concrete sand	Minus 5/16; fineness modulus 3
Plaster sand	Minus 3/16
Asphalt sand	Minus 1/8

All gravels are washed by means of sprays in the screens, the sand and water all going to the sand classifiers. Approximately 2,000 gallons of water per minute is used in this process, or 300 gallons per ton per minute.

Due to the surplus of sand, it is necessary to waste approximately 20 per cent, or 50 tons per hour, which was originally accomplished by a flume 2 feet wide by 2 feet deep, lined with $\frac{1}{4}$-inch steel plate, and carried on timber bents. This flume with an elevation at the sand drags of about 40 feet above the ground and an 8 per cent grade, discharged into a large pit about 300 feet west of the plant, which was opened up at the beginning of plant operations but was abandoned due to the high percentage of sand at this point in the deposit. Surplus sand was dumped here until the old pit was filled and the flume extended to a point where it met the fill and choked up. A receiving box 6 feet square and 10 feet deep was then constructed at a point in the flume where the suction for a sand pump could be taken from the bottom. An 8-inch direct-connected sand pump driven by a 100-hp., 750-r.p.m. motor was then installed. The discharge pipe line was run out horizontally from the pump for 10 feet, then a 45° elbow with 5-foot radius was installed and the line raised on this angle until it was 30 feet above the ground. It was carried at this height for 100 feet on timber bents, but as soon as the waste fill built up sufficiently and extensions were required, only short bents, 3 or 4 feet in height, were necessary to carry the line forward. The line was carried slightly down grade. This system worked successfully for about 800 feet, at which point another sand pump of the same design and size had to be installed as a booster. The discharge line at present extends approximately 250 feet beyond the booster pump. The sand pumps are equipped with white-iron liners which last about three months. The runners last about the same length of time.

Occasionally, due to trouble at the power plant, the voltage drops or the power is shut off altogether, causing considerable delays because the water velocity in the pipe line is not sufficient to carry the sand, which settles and plugs the line. When this happens it is necessary to clean the line out before starting. To facilitate this work a 3-inch pipe nipple is welded on the top of each 20-foot section of pipe and a 6-inch coupling and plug are welded to the bottom. With this arrangement a fire hose is connected to the 3-inch nipple and the 6-inch plug removed so that the sand is effectively washed out. However, at times it has been necessary to take the entire line apart and clean it out. The pipe sections are put together with flanges instead of screw joints. To prevent closing down the plant while the pipe line is being cleaned, a Y and valve were placed in the line near the pump and a short pipe line extended for emergency use. The cost of cleaning this pipe line is carried as a washing cost.

The pipe used is heavy casing and is turned one quarter of the way around every three months to distribute wear, and in this way it lasts for approximately one year.

Mention was made earlier in this paper of the contemplated disposal of surplus sand by flume to the present pit. The dragline previously described is now excavating that portion of the pit to be used for this purpose.

SAND CLASSIFIERS

The two sand classifiers operate on the intermittent-rake principle, the unclassified sand and water being fed into the deep end of a steel box having an inclined bottom. The course sands settle to the bottom of the box while the fine sands, silt, and lighter foreign particles overflow with the water. The settled sands are raked up the inclined bottom by a sand rake carried on bell cranks, the shafts of which are driven by spur gears of the same size, thus synchronizing the action of the head and tail cranks. These spur gears are driven by a pinion on a counter shaft which is operated by a cut-gear reduction from a $7\frac{1}{2}$-hp., 750-r.p.m. motor. The upper section of the rake is above the water level in the box, so that the clean sand has an opportunity to drain before being discharged. The rakes make 27 strokes per minute. Both machines are duplex and are arranged to produce coarse or concrete sand on one side and fine or plaster sand on the other. This is accomplished by cutting out a portion of the steel partition between the drags and bolting a 3/16-inch wire screen over this opening. The water and sand now flow to one side, and the agitation caused by the drag action forces a portion of the fine sand through the screen; the sand is then picked up by the drag rakes, pulled up the incline, and discharged into the storage bin.

The over-all size of the classifier boxes is 6 feet wide by 16 feet long and 3 feet deep. A steel partition down the center forms two 3-foot compartments for the drags. The rakes are made of $\frac{1}{2}$-inch steel plate 8 inches wide and 30 inches long and last approximately two years. The inclination of the box is 2 inches to the foot.

Due to a surplus of sand, a portion is sent directly to the waste flume. The two classifiers together discharge approximately 50 tons per hour of plaster sand and 150 tons per hour of concrete sand.

RECRUSHING METHODS

At times, due to unbalanced market conditions, there is an excessive demand for certain sizes which leaves a surplus of others, so it becomes necessary, after the storage piles reach capacity, to crush as much material as possible. This material for crushing is handled by a horizontal 24-inch belt conveyor running the full length of the bunkers near the bottom of the bins, each of which has a gate and short chute through which the material can be discharged onto the belt conveyor. The conveyor in turn discharges onto a 24-inch inclined belt conveyor with 260-foot centers leading to a point over the secondary crushers. The incline of 10° gives plenty of headroom to chute the material into any of the three secondary crushers. The 36-inch horizontal-disk crusher is generally used for this recrushing process.

The horizontal belt is driven by a $7\frac{1}{2}$-hp., 750-r.p.m. motor, and the inclined conveyor is operated by a 30-hp., 750-r.p.m. motor. Both conveyors travel 100 feet per minute.

Partly worn-out belts from this or some other plant of the company are used on these conveyors, as the work is light and intermittent on this unit.

STORAGE BUNKERS

The finished materials are chuted directly to bins, each having a capacity of 200 tons. There are 18 of these, nine on a side, giving a live storage capacity of 3,600 tons. Steel columns on concrete foundations are used as supporting members for the reinforced-concrete floor slab in the bins. The bin walls and partitions, as well as the superstructure, are of wood.

The gates on the bottom of the bins are 18 inches square and are made of cast iron. The quadrant which opens or closes each gate is operated from a platform along each side of the bunkers at such height as to enable the loader not only to watch the material as it discharges, but to observe if the car or truck body is clean. The opening and closing mechanism consists of an arm fastened to one side of the quadrant extending to a point above the loader's platform with an 18-inch rack on the end. The rack is geared to a small pinion and short shaft with a 12-inch chain sprocket on the other end. A short section of endless chain runs over this and hangs down to a point where it is convenient for the loader.

The loader has a form that he fills out whenever a car is loaded, giving the car number and the material contained. This form is given to the weighmaster at the close of the day to be checked against the bills of lading. This is an important procedure and practically eliminates wrong shipments, which in the past have entailed considerable cost for their disposal at a destination which might be 200 miles away.

LOCOMOTIVE CRANE

Material for ground storage is taken in carloads from the bunkers and unloaded by a steam locomotive crane. This machine is mounted on two standard 4-wheel railroad trucks with a 50-foot boom which carries a $1\frac{1}{2}$-yard clamshell bucket. The crane uses about seven barrels of oil per 10-hour day and moves and switches carloads for storage or shipment. It unloads about three 60-ton cars per hour, but its rate in loading from storage is somewhat faster.

SHIPMENTS

Shipments are made in cars, which are loaded directly from the bunkers under which there are double tracks, or in trucks which are loaded from the side of the bunkers. Carloads for commercial shipment are handled in the yard by a 60-ton steam locomotive with six drivers and tender, which brings the cars in from the empty tracks adjoining the railroad company's main line; after the cars are loaded and weighed, the same locomotive returns them to the loaded tracks to be switched out by the railroad.

All material is weighed at the plant in a scale house near the bunkers which is equipped with a 120,000-pound capacity railroad scale and a 40,000-pound capacity truck scale.

The Pacific Electric and Southern Pacific Railroad Companies serve as carriers of carload shipments.

STOCK PILES

Ground storage is used as a balance between production and shipments.

Approximately 60,000 tons of various materials can be piled in the ground storage system, the layout of which is shown in Figure 5. A perpetual inventory is kept showing the tonnage in each stock pile and is posted each day on a daily tonnage report by the weighmaster, who receives a report from the crane operator each evening. This report shows the number of cars loaded from stock pile for shipment, or unloaded for storage during the day and what material they contained. It then remains only to deduct from or add to the previous days inventory to bring it to date.

WATER SUPPLY

All water used in the plant is produced from two wells, both 500 feet deep, located alongside the plant. The casing of these wells is double-stovepipe, No. 8-gage steel, 16 inches in diameter. This is perforated with six $\frac{3}{4}$ by 3 inch slits per foot from a depth of 75 feet to the bottom of the wells, which perforations are made by a mechanical perforator after the drilling is finished and the casings set. The water level is approximately 100 feet below the tops of the casings.

Each well is equipped with an 8-inch, direct-driven, deep-well turbine pump, having a capacity of 1,500 gallons per minute, and maintaining a pressure of 60 pounds per square inch at the screens on top of the plant. Each of these pumps is driven by 125-hp., 1,000-r.p.m. vertical-type motor mounted on the pump head and frame. Water meters which are checked to determine the flow are set in the pipe lines to the plant, and their readings give warning when repairs are necessary. The superintendent has a chance to select the proper time for the repairs and to make arrangements to have the necessary material on hand, thus preventing a shutdown at an operating peak when it would cause the maximum difficulty.

POWER

Electric power, furnished by the Southern California Edison Co., is used throughout, except for the digging and transportation equipment, which is steam driven and uses oil for fuel. Sixty-cycle, 440-volt induction motors drive all crushers, screens, conveyors, hoists, etc. A survey of the power factor at this plant shows that theoretically the equipment is greatly over-powered, but owing to varying loads and the fluctuation in operating conditions in this type of work such provision of excess power is general practice. The average cost of power is $0.014 per kilowatt-hour.

SHOPS

The blacksmith shop is used only for minor repairing of plant equipment such as pit-car work, tool sharpening, making up of odd-length bolts, etc. One blacksmith and helper handle this work. All heavy repairs, maintenance, and building of any mechanical equipment is done in the Largo shops, which are the eastern-division headquarters of the Consolidated Rock Products Co. and which handle all this type of work for the nine plants located in this section.

TESTING AND SAMPLING OF MATERIALS

During the past few years the demand for better-type highway and building construction has increased rapidly in this section. Engineers have been making careful studies in design and have carried on extensive tests in order to prove or disprove their theories. This research has resulted in rigid specifications in the State and city departments of public works, so that it is necessary to carry on various tests of our materials at all times to safeguard against condemnation. These tests are made by a testing engineer in a laboratory centrally located around the plants at Largo. Various tests carried on for our records are screen analysis, dust content, silt and clay determination, void content, fineness modulus, specific gravity, weight per cubic foot, etc. A typical screen analysis showing uniformity of one type of product from 18 of our various plants is given below. This uniformity has been obtained only after much testing and experimenting with screen sizes and speeds.

Screen analysis of No. 3 gravel

Plant	No.	Per cent through				
		$1\frac{1}{4}$ inches	1 inch	$\frac{3}{4}$ inch	3/8 inch	$\frac{1}{4}$ inch
Alameda	1	99.5	90.0	71.0	12.0	2.0
Arroyo Seco	2	99.0	92.0	68.0	14.5	1.5
Durbin	3	98.0	89.0	74.1	16.5	2.0
Baldwin Park	4	98.0	90.0	72.0	15.0	2.0
Reliance	5	100.0	92.0	65.0	18.0	2.0
Irwindale	6	100.0	89.0	68.0	21.0	1.5
Builders	7	99.0	89.0	71.5	19.5	2.0
Kincaid	8	100.0	90.0	73.0	21.5	2.5
Rivas	9	100.0	89.0	71.0	19.0	2.0
Largo	10	100.0	92.0	73.5	21.0	1.5
Claremont	11	100.0	91.0	70.5	23.0	2.0
Orange Co.	14	98.5	90.0	72.0	20.5	1.5
Roscoe	22	100.0	96.0	69.0	29.0	2.5
Hewitt	23	100.0	97.0	76.0	24.0	2.0
Sheldon	24	100.0	96.5	65.0	19.0	1.0
Penrose 1/	25	100.0	88.0	68.0	19.0	2.0
Big T[illegible]	26	100.0	95.0	75.0	25.0	1.0
Boulevard	27	100.0	96.0	68.0	18.0	1.5

1/ Unwashed material.

Knowing that correctness in testing depends fundamentally upon the accuracy of samples, considerable time has been spent and experimenting done to achieve correct sampling, with the result that samples, even though taken by different men, now check closely. Samples from carloads of gravel are taken from at least 10 places throughout the loads at different depths to yield approximately 200 pounds or two sacks full. These are taken to the laboratory and quartered down to 50 pounds before drying and screening. Sand samples are taken in much the same manner, except that holes approximately $2\frac{1}{2}$ feet deep are dug in the load with a shovel, and then the point of the shovel is drawn up one side of the hole from the bottom to the top. Such samples are taken from at least six places in a carload, giving a total of 100 pounds or a sack of material, which is taken to the laboratory and quartered down to approximately 10 pounds. The sand is then dried and sampled down to 1,000 grams before screening.

SAFETY WORK

Safety work is under the direction of Fred L. Heth, safety engineer of the company, who has a safety committee at this plant composed of the plant superintendent, foreman, two men from the plant, and one man from the pit. A meeting is held each week and reports of these meetings are forwarded to the safety engineer. A general meeting of the division is held by the safety engineer each month, all members of the safety committee from the various plants being present. Safety work and recommendations are discussed and instructions in safety and first-aid are given.

Figure 6 shows the organization of the company.

PAY SYSTEM

The superintendent, foreman, clerk, and weighmaster are on a monthly salary. All other workmen are employed on an hourly basis.

	Wage
	Per month:
Superintendent	$300
Foreman	200
Clerk and weighmaster	150
	Per hour:
Shovel engineer	0.85
Fireman	.45
Craneman	.65
Locomotive engineer (top)	.65
Locomotive engineer (pit)	.60
Hoistman	.55
Millman	.50
Brakeman	.50
Loaders	.50
Blacksmith	.70
Crane operators	.65
Laborers	.35

TABLE 1.- SUMMARY OF COSTS

Period covered: Jan. 1, 1930, to Dec. 31, 1930.

Total material loaded during period: 702,888 short tons.

Operating costs per dry ton of sand and gravel mined

	Labor	Super-vision	Power	Fuel	Other supplies	Total
Mining						
Loading	$0.0149	$0.0015	$0.0004	$0.0104	$0.0029	$0.0301
Transportation ...	.0124	.0016	1/.0069	.0036	.0009	.0254
Miscellaneous	.0056	- -	- -	- -	- -	.0056
Crushing, primary ..	.0019	.0008	.0016	- -	.0010	.0053
secondary.	.0028	.0008	.0035	- -	.0019	.0090
Screening	.0031	.0008	.0021	- -	.0048	.0108
Conveying	.0030	.0008	.0021	- -	.0039	.0098
Washing	.0022	.0008	.0069	- -	.0009	.0108
Storage, in	.0100	.0025	.0005	.0019	.0006	.0145
out	.0067	.0025	.0004	.0014	.0004	.0114
Miscellaneous plant.	.0104	.0009	.0027	- -	.0019	.0159
Total operating ..	0.0730	0.0130	0.0271	0.0173	0.0192	0.1496

1/ Hoisting from quarry.

TABLE 2.- SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Period covered: January 1, 1930, to December 31, 1930.

Material loaded during period: 702,888 short tons.

Weight per cubic yard: 2,630 pounds.

	Mining	Crushing	Other	Total
Labor (man-hours per ton):				
Loading	0.0333	- -	- -	0.0333
Haulage	.0277	- -	- -	.0277
Miscellaneous	.0125	0.0124	0.0680	.0929
Supervision	.0053	.0026	.0102	.0181
Total labor	0.0788	0.0150	0.0782	0.1720
Average tons per man per shift	126.7	533.0	83.2	58.2
Labor, per cent of total operating cost				57.5
Power (kilowatt-hours per ton):				1.9378
Air compressors (loading)	0.0282	- -	- -	.0282
Crushing	- -	0.3643	- -	.3643
Screening	- -	.1510	- -	.1510
Hoisting 1/	- -	.4930	- -	.4930
Conveying	- -	.1510	- -	.1510
Washing	- -	.4930	- -	.4930
Storage	- -	- -	0.0643	.0643
Shops	- -	- -	.0620	.0620
Lighting	- -	- -	.1310	.1310
Fuel (barrels oil per ton)	0.0155	- -	0.0092	0.0247
Other supplies in per cent of total supplies and power	- -	---	- -	43.2
Supplies and power, per cent of total operating cost	- -	- -	- -	42.5

1/ This refers to transportation in Table 1.

TABLE 3.- DETAILED AVERAGE SHOVEL COSTS, DIRECT OPERATION

Period covered: Jan. 1, 1930, to Dec. 31, 1930.
Type of equipment: Railroad-type steam shovel.
Size of dipper: 2 cubic yards.
Tons of gravel loaded: 402,052.

Shovel operation	Amount	Cost per ton
Engineers	$1932.00	$0.0048
Cranemen	1320.00	.0033
Firemen	1158.47	.0029
Foremen	208.00	.0005
Pitmen	960.00	.0024
Other operating labor	60.78	.0001
Total operating labor	5639.25	0.0140
Fuel or power	3826.14	0.0094
Grease and lubricants	142.80	.0004
Other operating supplies	72.50	.0002
Total supplies	4041.44	0.0100
Repair labor	219.67	0.0005
Shop labor	130.50	.0003
Repair supplies	619.23	.0016
Shovel track	41.84	.0001
Total repairs	1011.24	0.0025
Total shovel operation	10691.92	0.0265

TABLE 4.- DETAILED AVERAGE DRAG-LINE COSTS, DIRECT OPERATION

Period covered: Jan. 1, 1930, to Dec. 31, 1930.
Type of equipment: Steam drag-line.
Size of scraper: 4 cubic yards.
Tons of gravel loaded: 300,836.

Drag-line operation	Amount	Cost per ton
Engineers	$2542.00	$0.0085
Firemen	1521.00	.0051
Foremen	208.00	.0006
Pitmen	905.58	.0030
Other operating labor		
Total operating labor	5176.58	0.0172
Fuel or power	2771.19	0.0092
Grease and lubricants	124.80	.0004
Other operating supplies	45.60	.0001
Total supplies	2941.59	0.0097
Repair labor	975.00	0.0032
Shop labor	333.59	.0012
Repair supplies	991.60	.0033
Shovel track	52.77	.0002
Total repairs	2352.96	0.0079
Total drag-line operation	10481.13	0.0348

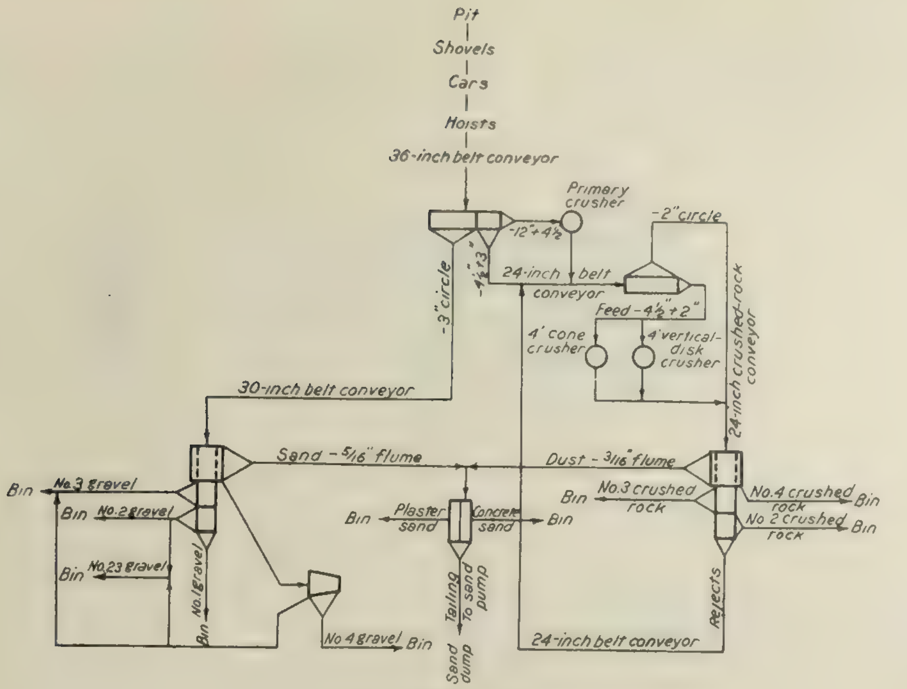

Figure 4.- Flow sheet

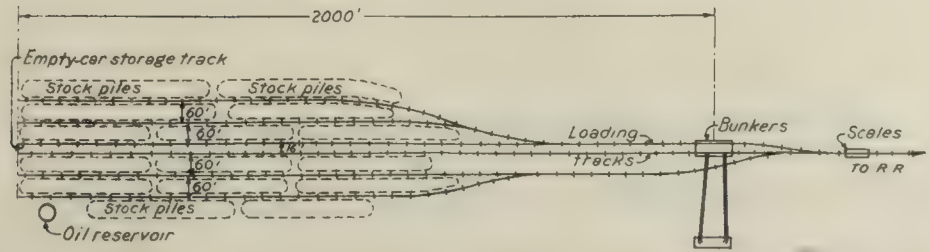

Figure 5.- Plan of ground-storage system

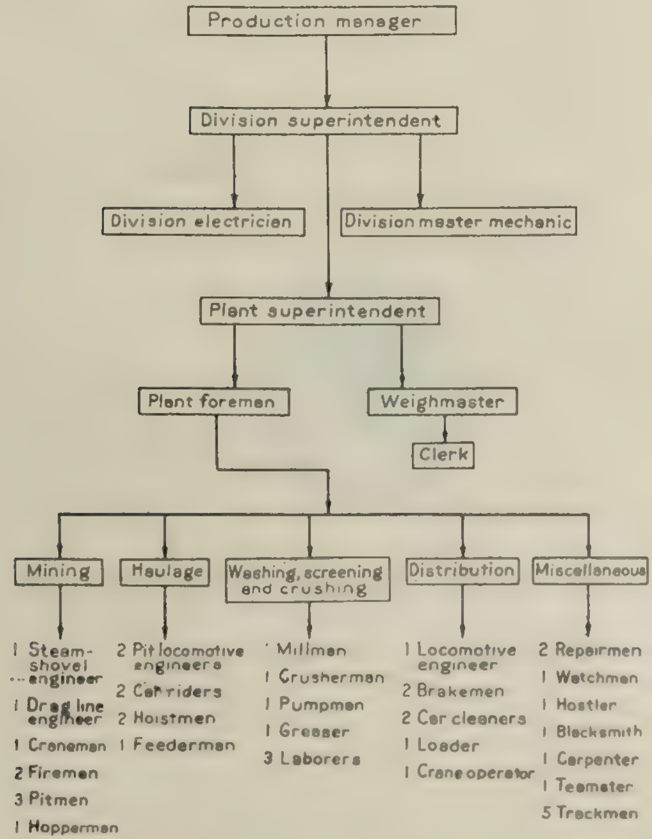

Figure 6.- Organization chart

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FEBRUARY, 1933

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

METHODS AND COSTS OF QUARRYING, CRUSHING AND GRINDING LIMESTONE AT THE PLANT OF THE SOUTHWESTERN PORTLAND CEMENT CO., EL PASO, TEXAS

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METHODS AND COSTS OF QUARRYING, CRUSHING AND GRINDING LIMESTONE AT THE PLANT OF THE SOUTHWESTERN PORTLAND CEMENT CO., EL PASO, TEXAS[1]

By Robert T. Mann[2]

INTRODUCTION

This paper is one of a series being prepared for and published by the United States Bureau of Mines on the quarrying, crushing, and grinding of limestone for cement manufacture. These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total costs. It is recognized that publication of total costs might in many instances cause embarrassment to individual producers, as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of the methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

ACKNOWLEDGMENTS

The author wishes to acknowledge the assistance of J. R. Thoenen, mining engineer of the Bureau of Mines, in the preparation of this paper.

HISTORY

The Southwestern Portland Cement Co. was organized about 1907 by Carl Leonardt. The property, which is about $3\frac{1}{2}$ miles northwest of El Paso on the Rio Grande River, was purchased from A. Courchesne, who was operating a limestone quarry in the district. Operations began about October, 1909.

At that time the rock was quarried adjacent to the crusher building and after being loaded by hand was hauled by wagon and mules only a short distance to the crushers. As unsuitable material was encountered in this deposit a new cut was opened about 1,000 feet further north which has developed into the present quarry.

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6608."

2 One of the consulting engineers, U. S. Bureau of Mines, and engineer, Southwestern Portland Cement Co.

During the plant's first years of operation almost its entire output was taken by the Elephant Butte project in New Mexico.

The two original kilns were 8 feet in diameter by 150 feet long. Later a third kiln was installed, and since then all three have been enlarged to 10 feet in diameter in the burning zone.

GEOLOGY

The rock in this district is said to belong to the Comanche series of the Lower Cretaceous period. It consists of massive gray and bluish limestones with interbedded calcareous shales of a total estimated thickness of 300 feet. There is practically no overburden, only a thin layer of weathered shale and limestone with a few scattered bowlders. Vegetation is also scarce and of desert variety, consisting mostly of cactus.

The topography is principally of foothill character, varying in elevation from 50 to 200 feet above the river. In the opencut of the present quarry the bench ranges from 80 to 140 feet above the quarry floor, which is only about 6 or 8 feet above the normal level of the Rio Grande; thus the quarry has natural drainage to the river and no water problems are encountered.

Just east of the plant site there is a deposit consisting entirely of yellow shale which is used occasionally when it is desired to mix material of low lime content without moving the electric shovel. Such a small amount is used that it is all loaded by hand.

PROSPECTING

There is no available data as to any test holes which may have been drilled to ascertain the extent of the cement rock deposits in the vicinity of the plant.

SAMPLING

All sampling is done in the blending building after the rock is crushed. The method used will be described in detail later.

CHOICE OF METHOD

The foothill topography of this vicinity, the elevation of the deposit in relation to that of the plant site, and the fact that no overburden is encountered, makes the opencut method of mining the only practical one to use.

MINING METHODS

Figure 1 shows a general plan of the quarry with track layout and the crushing, drying, and raw-grinding buildings. Figure 2 gives the chemical composition of the rock at various places in the face of the quarry.

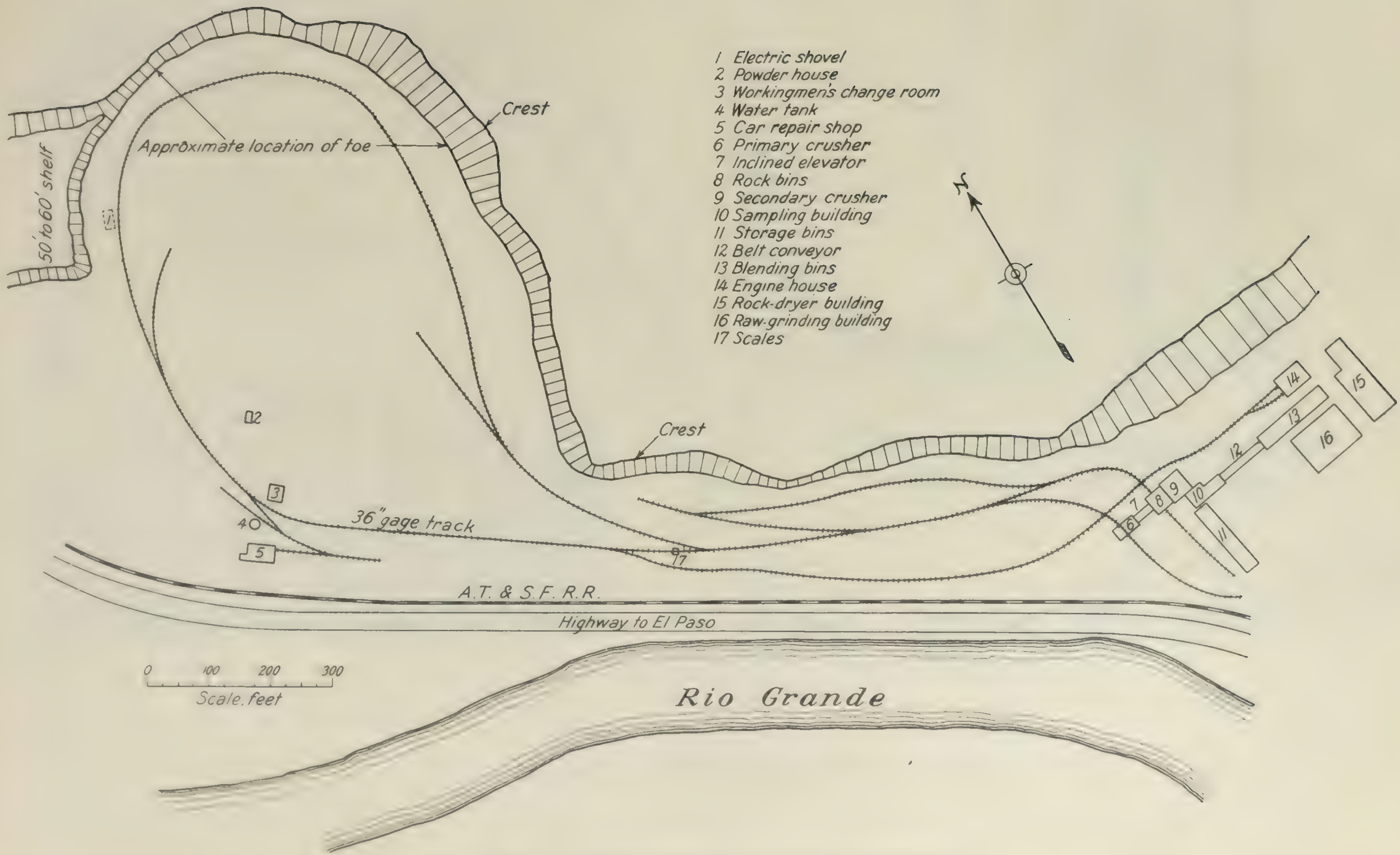


Figure 1.- Map of quarry and part of cement plant

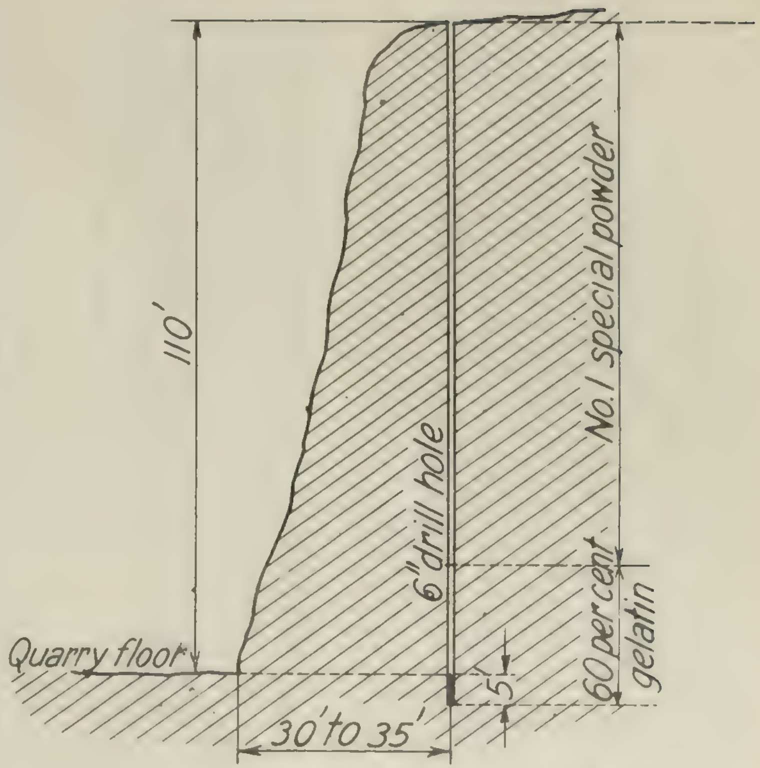
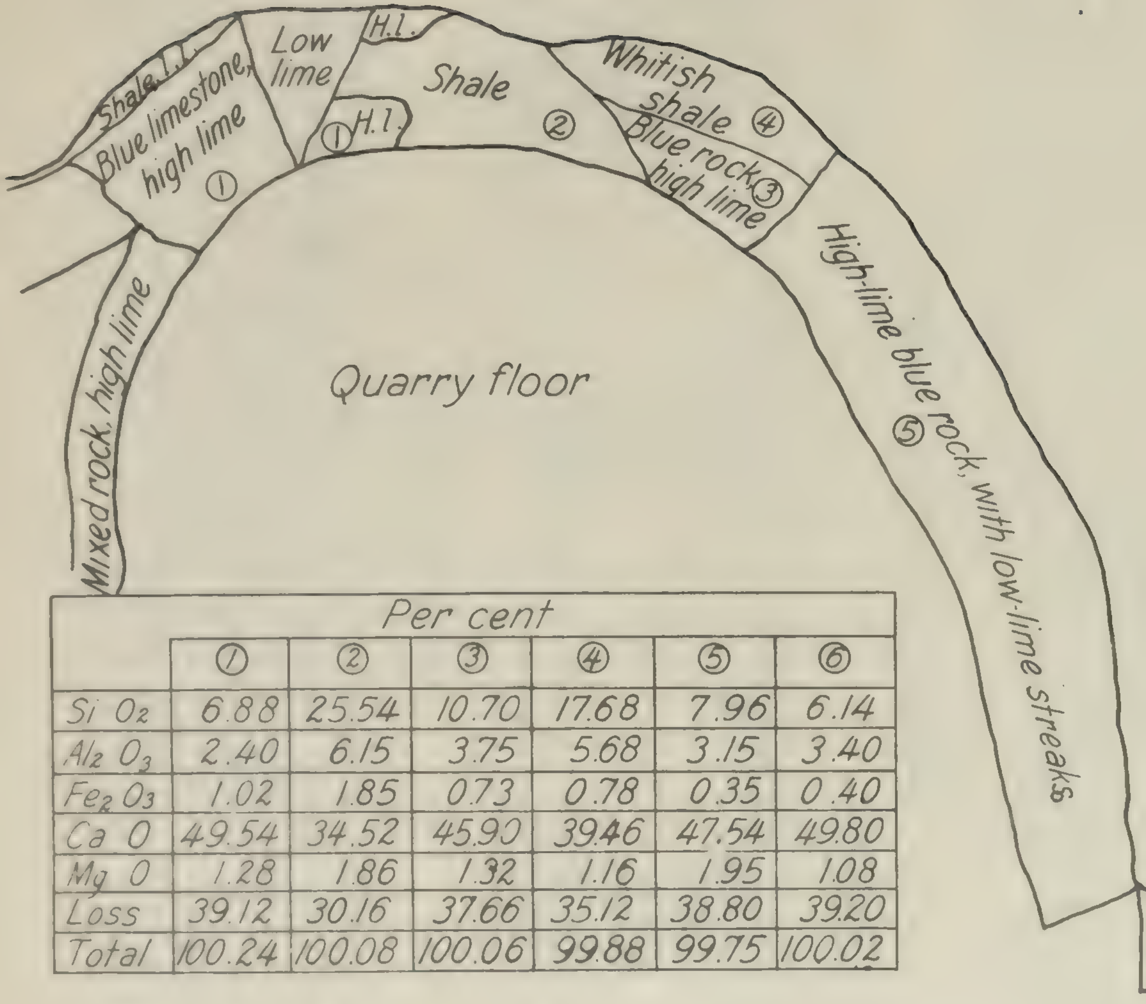
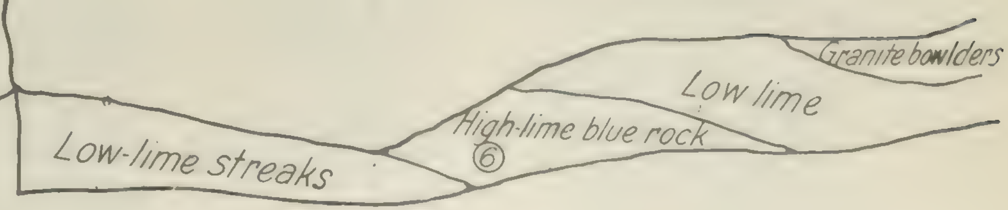


Figure 3.- Typical section of cut and method of loading drill holes

	Per cent					
	①	②	③	④	⑤	⑥
SiO_2	6.88	25.54	10.70	17.68	7.96	6.14
Al_2O_3	2.40	6.15	3.75	5.68	3.15	3.40
Fe_2O_3	1.02	1.85	0.73	0.78	0.35	0.40
CaO	49.54	34.52	45.90	39.46	47.54	49.80
MgO	1.28	1.86	1.32	1.16	1.95	1.08
Loss	39.12	30.16	37.66	35.12	38.80	39.20
Total	100.24	100.08	100.06	99.88	99.75	100.02

Figure 2.- Sampling map

During normal operation about 25,000 tons of rock per month is needed to supply the mill. This is all broken from the solid by large shots which bring down enough material to last four or five months. One shot fired in 1926 broke 350,000 tons, a yield of 5.68 tons of rock per pound of explosive used. However, this shot left the face in a dangerous condition and it required some time to straighten it out. At present, shots yield from 90,000 to 150,000 tons broken from the solid at one time. A typical section of the cut and the method of loading drill holes are shown in Figure 3.

PRIMARY DRILLING

All primary drilling is accomplished by means of a No. 4 traction well rig, powered by an 18-hp. gasoline motor. One man and a helper operate the rig. Drilling is contracted for at 30 cents per foot of hole drilled and cased when necessary. The drill helper is paid an hourly rate. The contract price includes only the operator's time, all supplies and repairs being furnished by the company.

Holes are drilled 6 inches in diameter with a 5 5/8-inch well drill bit. The rate of drilling depends on the hardness of the rock. In hard limestone the rate is about 4 feet per hour and in softer stuff the rate may be about 8 feet per hour. The drill holes are spaced 12 feet apart in rows of 20 or 30 holes parallel to the face of the cut. Each hole is 30 or 35 feet back from the toe and is drilled 5 feet below the quarry floor (see fig. 3). A record is kept of each hole by plotting its location on a large-scale map. Casing is used only in loose material.

Average cost of well drilling

Total feet drilled, 5,163   Period covered, 1930.

Item	Cost per foot	Cost per ton of rock broken
Repair labor	$ 0.06599	$ 0.00141
Repair parts	.02571	.00055
Operating labor ..	.46788	.00999
Drilling cable ...	.07607	.00163
Other supplies ...	.10249	.00219
Total	0.73814	0.01577

PRIMARY BLASTING

All primary loading and blasting is done under the direct supervision of an experienced powder man furnished by the powder company. A crew of six or eight men is employed in loading the holes for a shot, which requires about eight hours. Sixty per cent gelatin dynamite is used in the bottom of each hole to a height of about 20 or 25 feet. This is well tamped with a 3-inch diameter oak tamping stick 3 feet long lowered into the hole by a rope. The primer placed in the bottom of the hole is a 5 by 8 inch stick laced through

with No. 6 double-countered cordeau. The rest of the hole is loaded with No. 1 special bag powder. After all the holes are loaded the fuse projecting from each hole is split and well wrapped around the No. 6 plain cordeau main-line fuse on one end of which is placed an electric detonator. The circuit is then completed and the shot fired with an electric blasting machine. Good fragmentation is obtained and not much secondary blasting is necessary.

Over a period of three years an average for all explosives used, including that for secondary blasting, showed 4.35 tons of rock quarried per pound of explosive. For about half of this period the rock was hand loaded and therefore had to be blasted into sizes that could be handled by one man. During the latter half of the period, however, most of the loading was done by shovel, which necessitated little secondary blasting.

SECONDARY DRILLING AND BLASTING

About 10 per cent of all rock quarried requires secondary drilling and blasting. Jackhammers are used for drilling. Air at about 80 pounds is supplied to the hammers by compressors in the waste-heat building. Another compressor in the quarry car-repair shop may be used, if necessary.

There are two air compressors in the waste-heat building, but only one is in use at a time. Compressor No. 1 has a 12-inch stroke and $7\frac{1}{2}$ and 13 inch diameter cylinders. It has a capacity of 365 cubic feet per minute and is belt-driven by a 50-hp. motor at 175 r.p.m. Compressor No. 2 has a 12-inch stroke and 8 and 14 inch cylinders. It has a capacity of 530 cubic feet per minute and is belt-driven by a 75-hp. motor at 250 r.p.m. The compressor in the quarry shop has a 12-inch stroke and $7\frac{1}{2}$ and 10 inch cylinders. It has a capacity of 250 cubic feet per minute and is belt-driven by a 50-hp. motor.

Two men are employed as drillers and work eight hours per day. Drills are 7/8-inch hexagon steel with 1 and $1\frac{1}{2}$ inch bits.

The holes are loaded with 20 per cent gelatin dynamite in 7/8 by 8 inch sticks. No. 6 detonators are used to which are attached 3 feet of safety fuse with a burning speed of 1 foot per minute. This allows time to light several shots with safety.

LOADING STONE

An electric full-circle caterpiller-type shovel with 17/8-yard dipper is used in loading the stone into cars. Alternating current is supplied to the shovel through a 350-foot special armored cable. A 2,300-volt line direct from the power house is stepped down by 2,300 to 440 volt transformers placed about in the center of the quarry floor. One shovel operator and an oiler-helper are employed at the shovel, which loads about 950 tons of rock in an 8-hour shift. It is seldom necessary to operate for more than eight hours in one day.

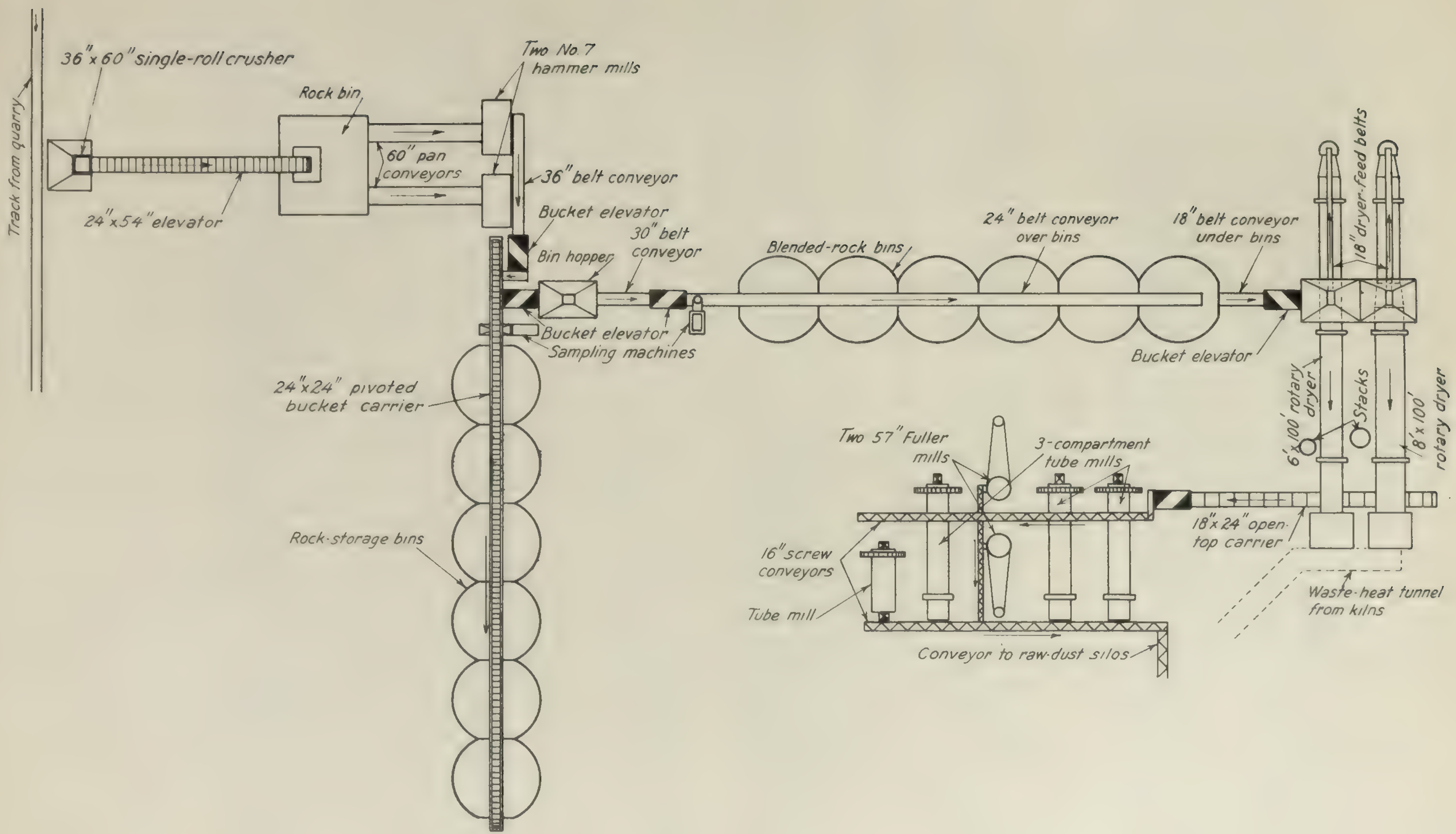


Figure 4.- Flow sheet of crushing, drying and raw-grinding departments

TRANSPORTATION

The rock is loaded into 6½-ton dump cars which are hauled to the crusher in trains of 10 by a 19-ton 36-inch gage steam locomotive. The average haul is about 2,000 feet. (See fig. 1.) For 800 feet just before reaching the primary crusher there is a grade of about 1.2 per cent. All other tracks in the quarry are nearly level. While one train of cars is on the way to and from the crusher, another train is being loaded at the shovel. The cars were formerly spotted by the shovel. However, this was discontinued recently, as it was found to damage the shovel boom and stick. At present the cars are spotted by a 12-ton gasoline locomotive.

CRUSHING PLANT

At the primary crusher the loaded cars are dumped by hand, using a pipe inserted in a dumping socket on the car and one on the floor. By pulling the car forward a little, one side is raised until the load dumps itself.

Figure 4 shows a flow sheet of the crushing, drying, and raw-grinding departments.

A 36 by 60 inch single-roll crusher operating at 39 r.p.m. receives any size feed up to 1-ton pieces, and reduces them to about 4 inches maximum size. The tonnage crushed in an 8-hour day averages about 100 tons per hour, but the crusher runs intermittently. Its maximum capacity is about 500 tons an hour. The crusher is belt-driven by a 150-hp., 440-volt motor.

Underneath the crusher a 4 ft by 14 ft 8-inch reciprocating plate feeder driven from the crusher receives the rock and delivers it into a bucket elevator inclined at 45° and having a vertical lift of 50 feet.

The elevator is of roller-chain construction with buckets 34 inches wide by 18 inches deep spaced 24 inches apart. It is driven 60 feet per minute by a 75-hp., 440-volt motor. This elevator, as well as the single-roll crusher, is operated only when there is a trainload of rock from the quarry. The discharge from this elevator falls into a small storage bin which is directly over two parallel 60-inch pan conveyor feeders that supply the two No. 7 hammer mills which crush to about ¾-inch maximum size. These are directly connected to two 150-hp., 440-volt motors operating at 690 r.p.m.

A 36-inch belt conveyor 32 feet 6 inches between centers and running 360 feet per minute takes the product from the hammer mills and delivers it to a 36-inch by 25-foot concrete-encased vertical bucket elevator of chain construction, driven 110 feet per minute by a 15-hp. motor. Buckets are spaced 18 inches apart.

This elevator discharges into the lower run of a 24 by 24 inch pivoted bucket carrier which is 428 feet long and serves six 700-ton concrete bins, either filling or emptying them. It is driven about 55 feet per minute by a 20-hp. motor.

The first sampling is done on the upper run of this carrier before it reaches the storage bins. (See fig. 5.) One bucket out of every 30 is tripped and discharged into a No. 1 pulverizer direct connected to a 20-hp. motor, operating at 1,120 r.p.m. The product of this mill, about $\frac{1}{4}$-inch in size, falls into a 26-inch Vezin sampler and is cut twice with 20° cutters. The final sample is taken hourly to the laboratory for an analysis which determines whether the rock shall go into the high-lime or the low-lime rock bins.

MIXING AND BLENDING

Due to the fact that the carrier buckets underneath the storage bins do not overlap it has been an awkward and dusty operation to draw rock from the bins. However, the crusher foreman has recently developed an automatic bucket feeder, which is operated by the carrier and which allows no spill between the buckets. (See fig. 5.) With this arrangement a small amount of rock may be drawn from each of the 6 bins at one time, thus blending the rock to a uniform mix. This installation eliminates the use of one man in the tunnel who previously filled the buckets and prevented as much spillage as possible.

The blended rock is now dumped into a hopper which is connected to the foot of a 30-inch by 70-foot vertical bucket elevator. This elevator is of chain construction, has a double row of 12-inch buckets spaced on 12-inch centers, and is driven 114 feet per minute by a 35-hp. motor through a countershaft. The elevator has a steel casing.

At this point the material was at one time dropped through a weighing machine. This machine, however, has since been taken out, and the rock merely goes through a hopper to a 30-inch horizontal belt conveyor running 105 feet per minute which delivers it to the foot of another 30-inch by 60-foot bucket-and-chain elevator operating at 120 feet per minute. The buckets are spaced on 12-inch centers.

From the top of this elevator the rock is chuted through another Vezin sampler where a 36° cutter takes part of the rock. The sample is cut again and then ground in another No. 1 pulverizing mill, from which it passes through another 26-inch Vezin sampler for the final sample, as previously described.

The main portion of rock discharges on a 24-inch belt conveyor which serves the full length of six 700-ton blended-rock bins. Of this belt conveyor the first 106 feet are inclined at an angle of 8° 30'; the rest is horizontal. Its total length between pulleys is 236 feet and it travels 398 feet per minute.

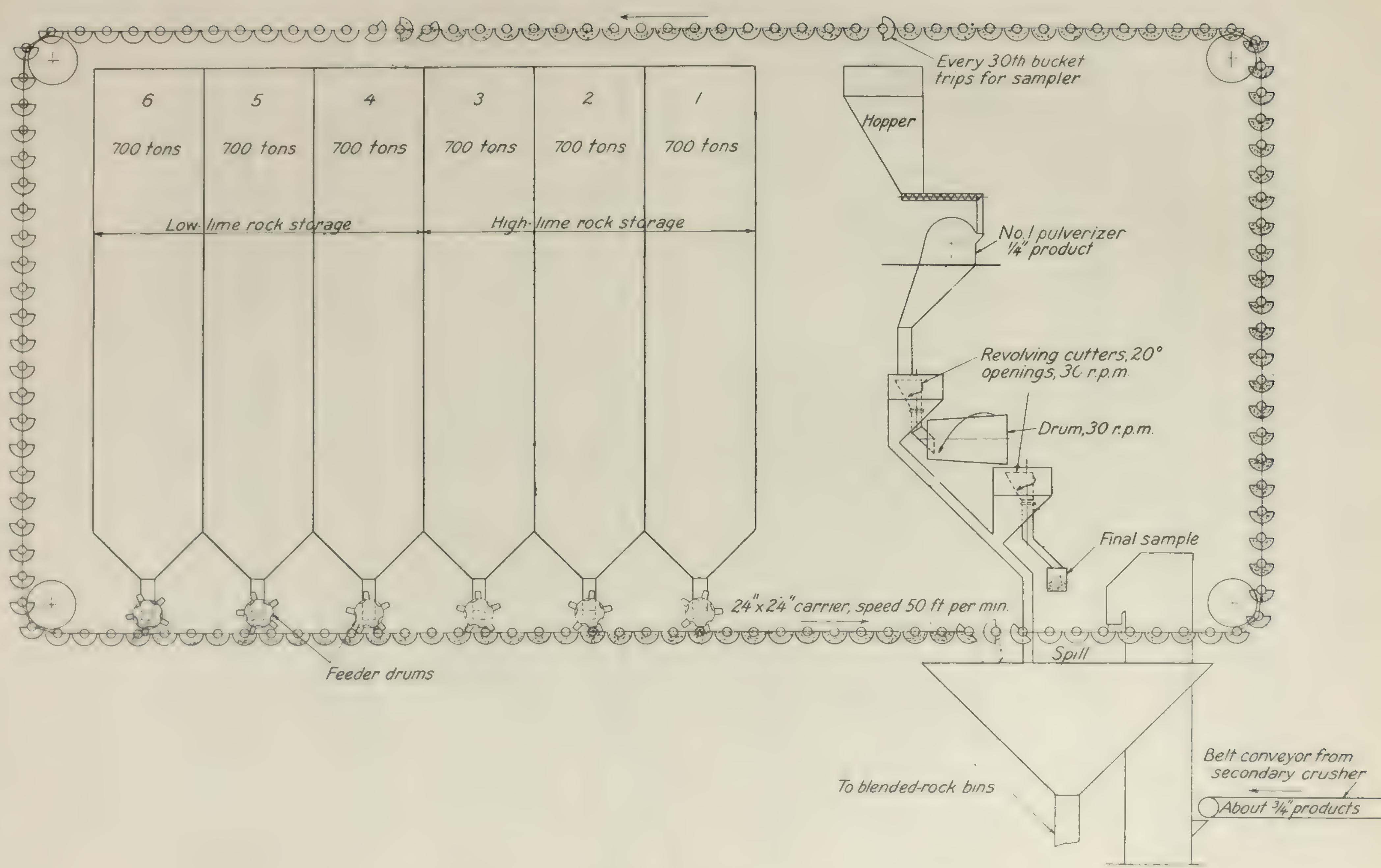


Figure 5.- Diagram of method of sampling and blending before drying

The rock is further blended by distributing the discharge of the conveyor to several bins by means of an automatic belt tripper traveling 50 feet per minute.

The two elevators just described and the belt conveyors are all driven from the same countershaft.

Three men and a foreman are employed in the crushing and blending departments, who work eight hours per day.

DRYING

Underneath the blended-rock bins is an 18-inch belt conveyor, 254 feet between centers and driven 250 feet per minute by a 5-hp. motor. The first 100 feet of this conveyor is inclined at an angle of 5°, the rest of it being horizontal. It discharges to the foot of a 24-inch by 58-foot vertical chain-and-bucket elevator driven 98 feet per minute by a 10-hp. motor. The buckets on this elevator are spaced on 12-inch centers.

The elevator discharges directly into the hopper over No. 1 dryer. In order to get material into the second hopper the first must be nearly full, as the rock flows from one to the other through a hole in the partition. For this reason No. 2 dryer is not operated as much as No. 1.

These hoppers are directly over each dryer, but are 37 feet from their feed ends so that an 18-inch belt conveyor under each hopper is used to carry the material to the ends of the two dryers. Each conveyor is inclined 1 inch in 12 inches, runs 90 feet per minute, and is 43 feet between centers.

The two dryers are each 100 feet long. One is 6 feet in diameter throughout its entire length and has a capacity of about 35 tons an hour. It is driven by a 50-hp. motor. The other is also 6 feet in diameter but has an enlarged section 8 feet in diameter at the lower end. It has an average capacity of 50 tons an hour and is driven by a 75-hp. motor. Each dryer is inclined $\frac{3}{4}$ inch per foot and makes 2.1 r.p.m.

The rock may be dried either by burning natural gas or by using part of the waste-heat gases from one of the kilns. However, when only one kiln is operated all the waste heat is needed for power generation.

After the rock is dried by heating it to about 325°F., it is received by a horizontal bucket conveyor of roller-chain construction, 192 feet long and having 24 by 18 inch buckets spaced 24 inches apart. The conveyor travels 32 feet per minute and delivers the dried rock to the raw-grinding building. One man on each shift is employed in this department.

Each unit in the crushing plant up to the blended-rock bins handles about 100 tons an hour. Beginning with the belt conveyor beneath the blended-rock bins each unit handles about 38 tons per hour.

RAW GRINDING

In this department the dried material is distributed to the various mill bins by means of a 24-inch by 56-foot steel-cased vertical bucket elevator operating at 120 feet per minute and with buckets spaced on 12-inch centers, and a 16-inch screw conveyor 62 feet long turning 100 r.p.m. Both are driven by one 20-hp. motor. Each unit handles 38 tons per hour.

There are six grinding units for pulverizing the raw mix. Three of these units are 6 by 36 foot Luther compartment ball mills which have a capacity of about 12 tons per hour and each of which is driven through a magnetic clutch by a 350-hp. synchronous motor. Two units are 57-inch Fuller-Lehigh mills each driven by a 150-hp. motor, and the other unit is a 6-foot, 6-inch by 20-foot Smidth ball mill which has an average output of 12 tons an hour and is driven by a 250-hp. motor. The Smidth mill is used to regrind the rejects from a 10-foot diameter mechanical air separator which treats the discharge from one of the Luther mills and the two Fuller mills.

GRINDING DATA

No. 1 Luther Mill

Lining .... Chrome-nickel steel.
Balls .... Heat-treated chrome-nickel steel in first and second compartments.
Danish pebbles in third compartment.
Ball sizes ..... $1\frac{1}{4}$ to $3\frac{1}{2}$ inches.
Ball load ...... First and second compartments .... 48,500 pounds
Third compartments .... 22,000 "
Total load .... 70,500 "

Liner wear ..... 0.22 lb. per ton ground.
Ball wear ...... No data.
Speed .... $24\frac{1}{2}$ r.p.m.

Screen Analysis

Screen size	Feed passing screen, per cent	Discharge passing screen, per cent
1 inch	100.0	-
3/4 do.	93.6	-
3/8 do.	83.4	-
4 mesh	61.2	-
8 do.	43.8	100.0
14 do.	28.0	99.4
28 do.	18.0	99.1
48 do.	11.0	99.0
100 do.	5.4	94.8
200 do.	-	82.2

No. 2 Luther Mill

Lining and load, same as No. 1 Mill

Screen Analysis

Feed	Screen size, mesh	Discharge passing screen per cent
Same as for No. 1 mill.	8	100.0
	14	99.4
	28	98.6
	48	98.0
	100	94.0
	200	83.0

No. 3 Luther Mill

Lining .... First and second compartments ... chrome-nickel steel.
Third compartment ... quartzite blocks.

Ball load and sizes - same as No. 1 mill.

Liner wear ..... No data.
Ball wear ...... No data.
Speed .... 24-1/2 r. p. m.

Screen Analysis

Feed	Screen size, mesh	Discharge passing screen per cent
Same as for No. 1 mill.	4	100.0
	10	98.4
	20	98.3
	30	98.0
	100	94.0
	200	79.8

Total cost for new balls for three Luther mills in 1930 was $25.

No. 4 Fuller Mill

Four balls ..... $15\frac{1}{4}$ inches in diameter.
Speed .... 127 r. p. m.
No data available on screen analyses of feed and discharge.

No. 5 Fuller Mill

Same as No. 4 Fuller mill.

The two Fuller mills have used 14 new balls each during the past eight years.

No. 6 Smidth Mill

Lining .... Quartzite blocks.
Balls .... $\frac{3}{4}$ and 7/8 inch steel.
Load .... 22,000 pounds.
Speed .... 24 r. p. m.

Screen Analysis

Screen size, mesh	Feed passing screen, per cent	Discharge passing screen, per cent
4	100.0	100.0
10	92.6	99.98
20	92.5	99.88
30	91.9	98.0
100	80.8	86.80
200	68.1	

The Smidth mill has been in operation for two years without repair to liners or addition of pebbles.

An average screen analysis for the discharge from six units shows 82 per cent through 200 mesh and 97 per cent through 100 mesh.

The discharge from the raw-grinding mills is taken by screw conveyors and elevators to raw-mix silos, where it is further blended before going to the kilns.

There are two men employed in the grinding department on each 8-hour shift, the mills operating 24 hours a day.

PER CENT EXTRACTION

All the rock quarried is used in cement manufacture, except a small amount of the raw dust ($\frac{1}{4}$ of 1 per cent) which is sold for rock-dusting in coal mines.

PAY SYSTEM

All wages are paid on an hourly basis except those of a few contractors who are employed part time in loading shale.

Hourly wage scale

Blacksmith	$0.69
Shovel operator	.80
Powder man	.58
Locomotive operators	$.55\frac{1}{2}$
Locomotive brakeman	.28
Drillers	.30
Dryer operators	$.25\frac{1}{2}$
Crusher operators	$.32\frac{1}{2}$
Millers	.35
Helpers and laborers	.23

SAFETY METHODS

During recent years much improvement has been made in safety work at this plant. On June 30, 1931, the quarry and transportation crews had worked 840 days without a lost-time accident. The crusher had operated 1,814 days, the dryer 752 days, and the raw-grinding department 2,009 days, all without a lost-time mishap. A meeting is held once a month for all foremen, in which safety problems are discussed. All the employees have received the course in first-aid training given by the U. S. Bureau of Mines.

A bonus of 1 per cent of regular pay is given each month to the employees in each department that operates the full month without a lost-time accident.

GENERAL

The locomotives use about 0.4 pound of coal per ton of rock hauled.

The average fuel consumption in the dryers is about 138 cubic feet of gas per ton dried.

Figure 6 shows the organization chart of the company.

COSTS

Period covered: January 1 to December 31, 1930.

Production of stone during period: 242,320 short tons.

Table 1. - SUMMARY AND DISTRIBUTION OF COST

Cost per dry short ton of stone

Department	Labor	Supplies	Power	Fuel	Explosives	Total
Quarry	$0.1000	$0.0260	$0.0015	$0.0011	$0.0255	$0.1541
Transportation	.0312	.0217	---	.0044	---	.0573
Crushing and sampling	.0291	.0232	.0048	---	---	.0571
Drying	.0302	.0100	.0044	.0251	---	.0697
Total	0.1905	0.0809	0.0107	0.0306	0.0255	0.3382

Table 2. - QUARRY COSTS

Cost per dry short ton of stone

Item	Labor	Supplies	Explosives	Power	Coal	Total
Well drilling	$0.0114	$0.0044	---	---	---	$0.0158
All blasting	.0061	.0004	$0.0255	---	---	.0320
Secondary drilling	.0185	.0027	---	---	---	.0212
Loading 1/	.0395	.0135	---	---	.0011	.0541
Loading tracks ...	.0071	.0018	---	---	---	.0089
Power, light, air and water	.0052	.0027	---	2/$0.0015	---	.0094
Supervision	.0117	.0002	---	---	---	.0119
Buildings	.0005	.0003	---	---	---	.0008
Total	0.1000	0.0260	0.0255	0.0015	0.0011	0.1541

1/ Because part of the stone was loaded by hand, as noted below, these figures do not check with those shown as shovel costs in Table 6.

2/ This is the power cost of the electric shovel per ton of all rock quarried. However, the shovel loaded only 188,341 tons of this amount. The other 53,979 tons was loaded by hand by contractors.

Table 3. - TRANSPORTATION COSTS

Cost per dry short ton of stone

Item	Labor	Supplies	Coal	Total
Locomotives	$0.0190	$0.0089	$0.0044	$0.0323
Cars	.0077	.0088	---	.0165
Tracks and roadway	.0043	.0038	---	.0081
Power, light, air and water	.0001	.0001	---	.0002
Buildings	.0001	.0001	---	.0002
Total	0.0312	0.0217	0.0044	0.0573

Table 4. - CRUSHING AND SAMPLING

Cost per dry short ton of stone

Item	Labor	Supplies	Power	Total
Crushing and sampling	$0.0198	$0.0185	---	$0.0383
Elevating and conveying ...	.0048	.0037	---	.0085
Power, light, air and water	.0030	.0003	$0.0048	.0081
Buildings and bins	.0015	.0007	---	.0022
Total	0.0291	0.0232	0.0048	0.0571

Table 5. - ROCK-DRYING COSTS

Cost per dry short ton of stone

Item	Labor	Supplies	Power	Fuel	Total
Dryer operation	$0.0176	$0.0056	---	$0.0251	$0.0483
Elevating and conveying ...	.0057	.0011	---	---	.0068
Power, light, air and water	.0051	.0021	$0.0044	---	.0116
Buildings and bins	.0018	.0012	---	---	.0030
Total	0.0302	0.0100	0.0044	0.0251	0.0697

Table 6. - AVERAGE SHOVEL COST

Period: 1930. Amount loaded: 188,341 tons.

Type 37 electric shovel. Dipper capacity: 1-7/8 cubic yards.

Item	Cost per ton
Repair labor	$0.01219
Repair parts	.01114
Total repairs	.02333
Operating labor	.01158
Operating supplies	.00124
Total operating	.01282
Power	.00190
Total cost	0.07420

Table 7. - SUMMARY OF COSTS IN UNITS OF LABOR AND POWER

Department	Man-hours per ton	Tons per man-hour	Kilowatt-hours per ton
Quarry	0.236	4.23	0.48
Transportation.	.073	13.60	----
Crushing	.086	11.7	1.58
Drying	.081	12.3	1.45
Grinding	.175	5.7	23.46
Total	0.651	-----	26.97

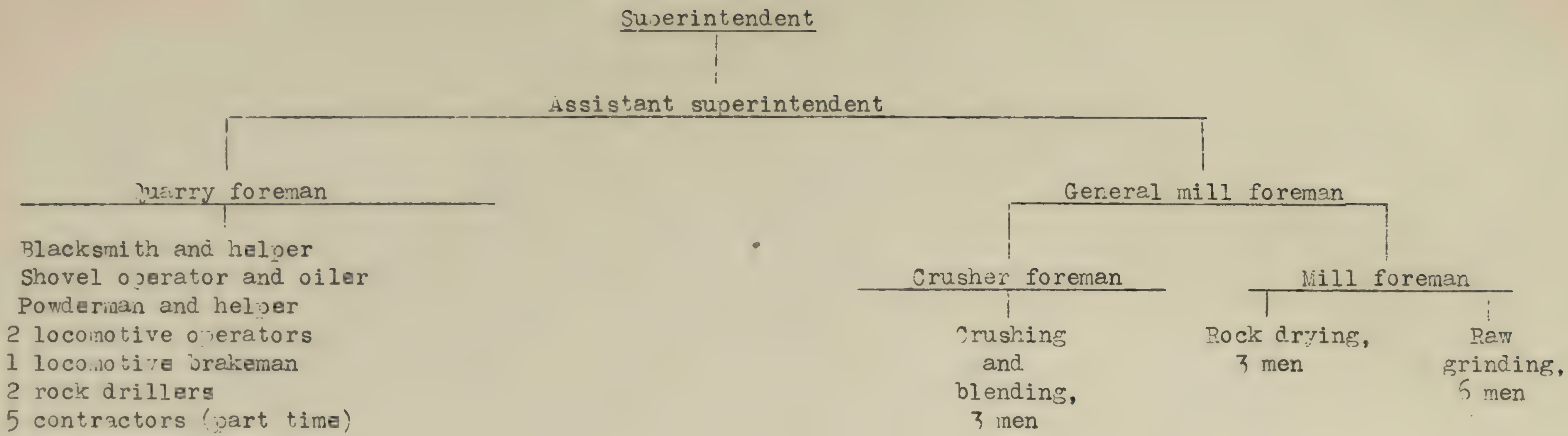


Figure 6.- Organization chart

I. C. 6609 July, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

Quarrying and Crushing Methods and Costs at the Santa Catalina Island Quarry of Graham Bros. (Inc.), Santa Catalina Island, Calif.

BY

GEO. ADAMS ROALFE

Information Circular
DEPARTMENT OF COMMERCE — BUREAU OF MINES

Quarrying and Crushing Methods and Costs at the Santa Catalina Island Quarry of Graham Bros. [Inc.], Santa Catalina Island California*

By Geo. Adams Roalfe†

Introduction

THIS PAPER is one of a series being prepared for and published by the United States Bureau of Mines describing methods of mining and crushing and the costs of operation at commercial crushed-stone plants throughout the country. These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total costs. It is recognized that publication of total costs might in many instances cause embarrassment to individual operators, as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of the methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

Acknowledgments

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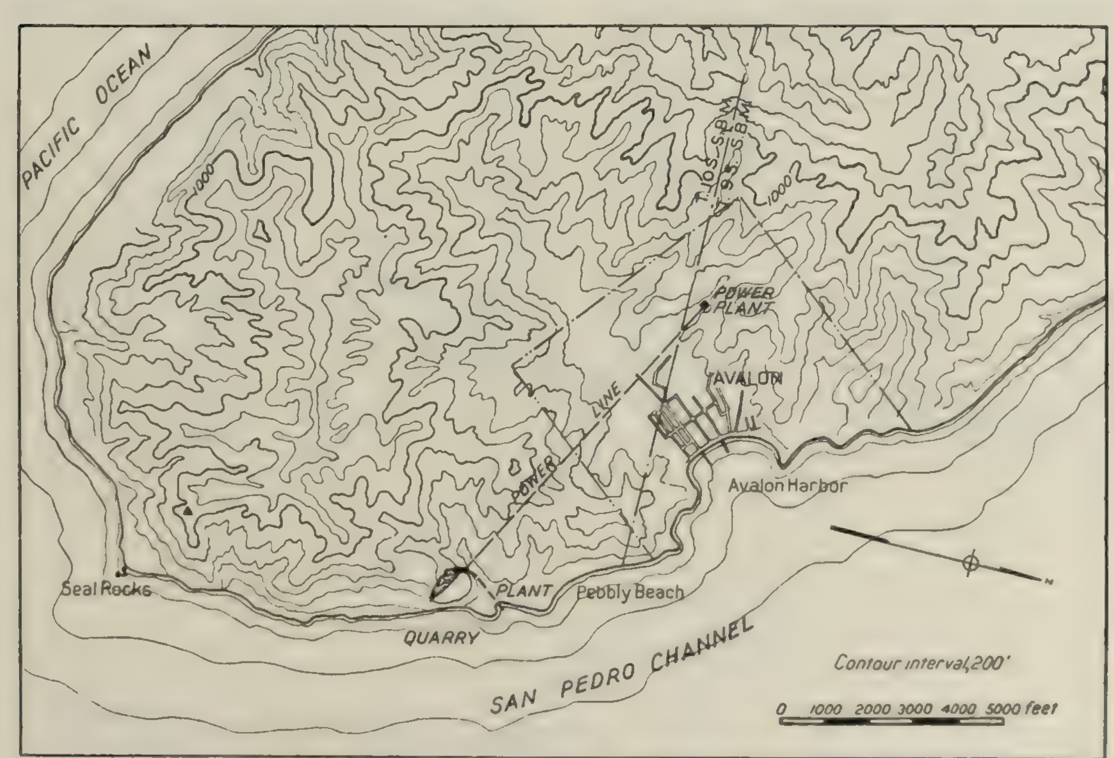

Fig. 1. Topographic map of portion of Santa Catalina Island

*The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6609."

†One of the consulting engineers, U. S. Bureau of Mines, and consulting engineer to Graham Bros. (Inc.).

the information presented in this paper. James Forrest, quarry superintendent, and R. D. Dillree, quarry foreman, fully cooperated in the collection of data.

Brief History and Early Development of Quarry

Prior to 1924, Graham Bros. (Inc.) had operated as general contractors, gradually entering the field as material distributors. Up to this time their production was limited to that from a sand pit on Signal Hill, adjacent to Long Beach. Other deliveries of aggregates were made from purchased production. In 1923 William Wrigley, Jr., owner of Santa Catalina Island, became interested in the business, and it was in this year that the Santa Catalina Island quarry was opened and the crushing plant constructed. Demands for material even before the plant was completed were such that a temporary gravel recovery plant was set up at Pebbly Beach adjoining the quarry site. The gravel from this plant was transported by barges to the mainland at Long Beach. The topography of that portion of Santa Catalina Island which includes the quarry and the townsite of Avalon is shown on Fig. 1. Fig. 2 shows the route taken by the barges and the relation of the quarry to the Long Beach distribution plant. The opening of the quarry and construction of the plant was carried out under the direction of Paul Graham, whose idea had led to the selection of the Santa Catalina site for the operation. The plant was formally opened and placed in operation on January 19, 1924, and has been running successfully since that date. The activities of Graham Bros. (Inc.) have expanded since 1924 until they now operate seven plants and distribute materials from two others.

Geology

Santa Catalina Island lies about 25 miles to the southwest from Long Beach. It is approximately 20 miles long and about 5 miles wide at its widest point. The rocks which compose it are almost entirely igneous. Their formation was the result of at least two major lava flows, one of which is exposed over most of the southern end of the island. The rock has been classed as andesite and is the material quarried. The topography is rough. Steep cliffs, intersected by precipitous canyons which lead to the ocean, rise abruptly from sea level. A few canyons extend out under the sea, making small harbors or bays.

The quarry was opened on a ridge forming a side of one of the less precipitous canyons. This location was selected because of the availability of good stone close to the beach and its accessibility from the city of Avalon, the largest community on the island.

As is true of most igneous rocks in this

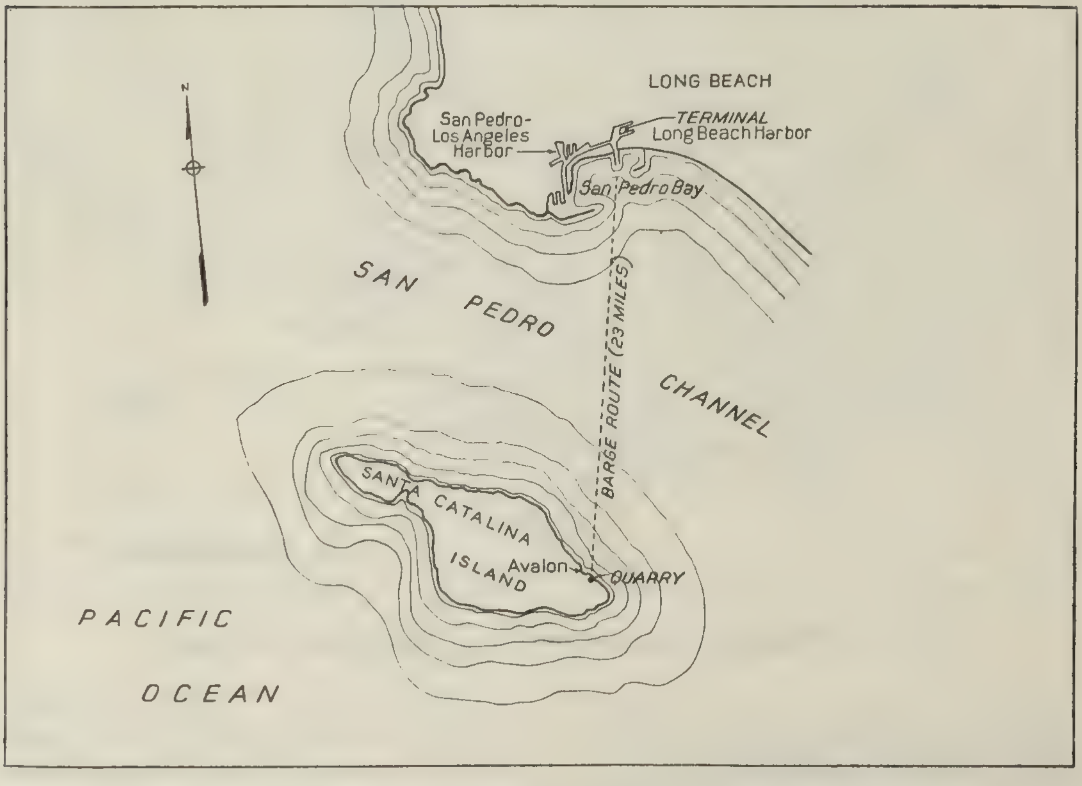

Fig. 2. Map showing rock-barge route between Santa Catalina Island quarry and Long Beach, California

region, the andesite is heavily fractured. In this particular area the overburden of soil is light. On the ridges it averages only a few inches, and even in the bottom of the gullies it is rarely over 3 ft. thick. From this it is seen that overburden has not been a particularly heavy problem.

In appearance the rock varies in color from a light buff through gunmetal blues to dark bottle-green. In the large unfractured masses the buff-colored material lies next to the seams and merges to either of the other types in the center of the mass. The buff-colored rock has been partly altered and is not as valuable as the darker material; however, it will successfully stand critical comparison with other rocks used commercially in this territory. The product of the quarry is a mixture of the various types noted and results in an excellent aggregate.

Clay is found filling the seams. This is largely an alteration product of the feldspar in the rock. When the quarry was first opened much material was wasted in disposing of this clay. Research disclosed that it made an excellent material for crusher-run road base. In grading it approaches the ideal for waterbound macadam and contains approximately 10% of fine clay thoroughly disseminated through the mass. If it were possible to balance the demand for crushed rock and quarry waste with the quantities produced it would be unnecessary to waste any material. During the calendar year 1930 nearly one-third of the tonnage sold was crusher-run base.

Physical Characteristics

The variation in color of the rock has been described. Tests to determine the compressive strength of the stone give the following results:

CRUSHING STRENGTH, LB. PER SQ. IN.	
No. 1 dark blue	14,615
No. 2 green	23,010
No. 3 buff	7,295
Average	14,973

The test specimens were 2-in. cubes. The average specific gravity of the types shipped is 2.63.

Aggregates in this territory are tested for abrasion resistance and toughness by the Los Angeles rattler test. The Los Angeles rattler consists of a cylindrical drum 28 in. in diameter and 20 in. long, mounted horizontally on a shaft and having a shelf 4 in. wide extending from end to end inside.

The charge consists of 11 lb. (5 kilograms) of aggregate of such size that all passes a 2-in. circular opening, 60% is retained on a ¾-in. circular opening, and all is retained

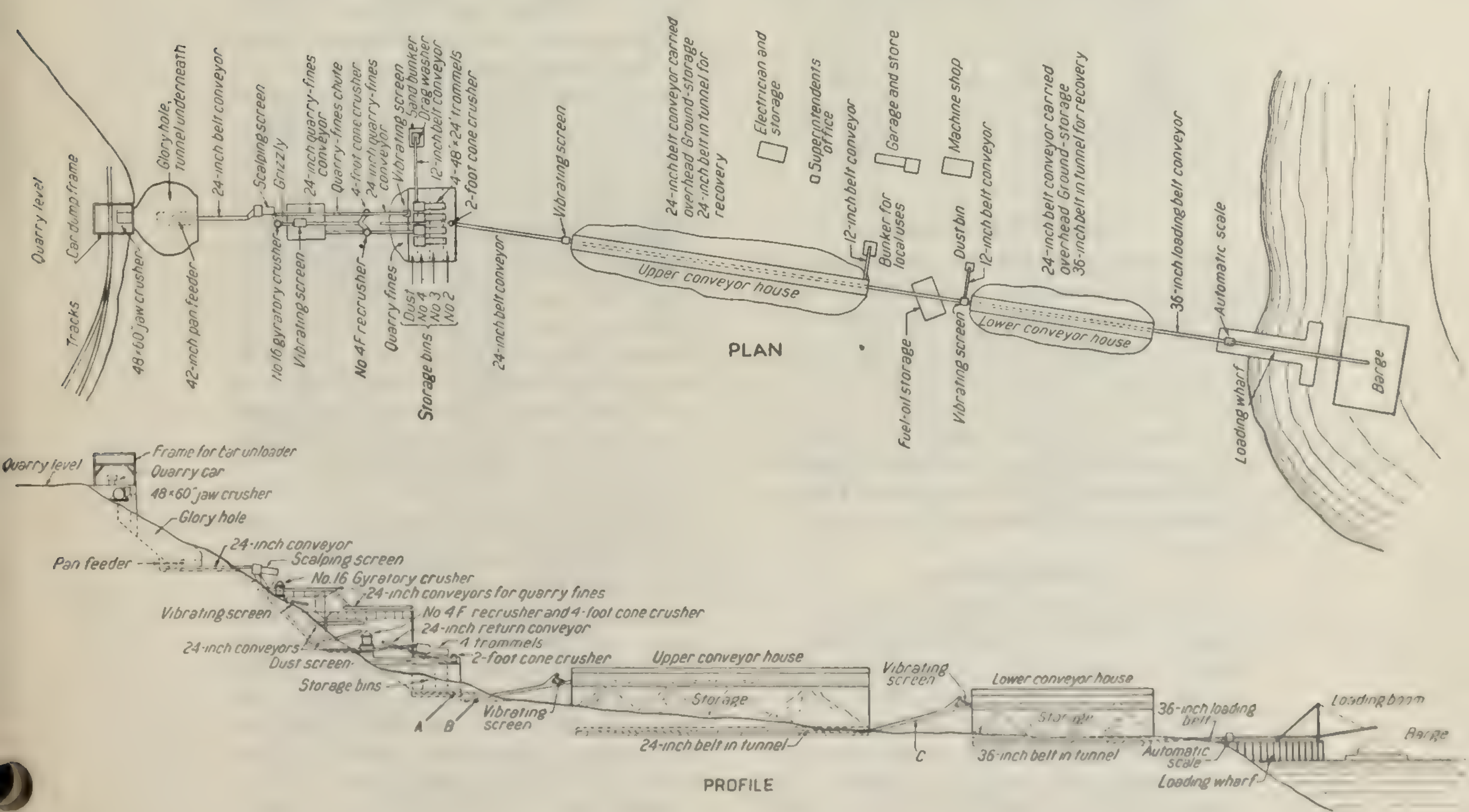

Fig. 3. Plan and profile of plant

on a ½-in. circular opening, together with 14 cubical blocks of cast iron having rounded corners and edges and weighing a total of 11 lb. After charging, the drum is turned between 28 and 30 r.p.m. The loss is reported as the per cent. of the sample which will pass a 10-mesh sieve after 100 and 500 revolutions, respectively.

Santa Catalina crushed stone when subjected to this test gives the following average results:

Revolutions	Loss, per cent.
100	4.24
200	10.38
300	12.66
400	14.20
500	19.20

Maximum losses permitted under local specifications vary from 30 to 50%. Data are not available for comparison with other standards.

The chemical composition of the rock has not been an important factor other than in determining its resistance to destruction when used as an aggregate in fireproof concrete. The stone falls in class A for this use. A chemical analysis of the commercial rock follows:

	Per cent.
SiO_2	61.05
Al_2O_3	18.30
Fe_2O_3	3.49
FeO	1.11
MgO	2.59
CaO	7.75
Na_2O	4.06
K_2O	1.36
MnO	Trace
TiO_2	0.09
P_2O_5	Trace
	99.80

Method of Exploration

Due to the nature of the deposit, unlimited tonnages are available. Prospecting of outcrops enabled satisfactory conclusions concerning persistence and quality to be reached, as the deposit is exposed not only at the top but also in the adjoining canyons. As most of the area under consideration is entirely free from overburden, the choice of location was most influenced by factors favorable to the building of the plant and to handling the product.

Mining Methods

The site selected adjoined a wide canyon opening on the beach and was about 200 ft. above sea level. Early work was done close to the plant and gradually extended around the ridge. One of the considerations which led to moving southward was the discovery of silver-lead-zinc ore in a fracture in the original quarry opening. This ore was worked by the Santa Catalina Island Co. for a number of years until the price of the metals fell below a profitable figure. At present work is carried on principally at the southerly end of the quarry, although the face has been carried back along its entire length from time to time. Since opening the quarry, approximately 2,500,000 tons have been handled. The quarry face now has a length of approximately one-half mile and averages about 400 ft. in height with a maximum of over 500 ft. The only preliminary development work required consisted of leveling a bench of suitable dimensions. As the quarry is an open-face operation, it has been possible to carry a level floor so that the trackage problem has been simple.

Figs. 1, 3 and 4 show the location of the quarry with reference to the crushing plant, a plan and profile of the plant, and the track layout. Due to the elevation of the quarry floor above the crusher plant and sea level, the disposal of waste material has been easily accomplished.

Stripping

The limited amount of overburden has been mentioned. On one or two occasions it was considered advisable to remove the soil, which averaged less than 2 ft. in thickness. This was done with teams and scrapers.

Drilling and Blasting

The quarrying methods in use vary little. All major shots are made with coyote holes at the quarry floor level and well holes at the top. Fig. 5 gives a typical coyote-hole layout and Fig. 6 a cross section of the quarry face showing the relation of the coyote hole to the top holes. In driving a coyote hole a tunnel is first driven at right angles to the quarry face to a depth of 50 ft. From this two crosscuts are made parallel to the face, each ordinarily 45 ft. long. In a coyote hole of the usual type pockets to carry the explosive are located 15 ft. each side of the entrance tunnel and at the ends of the crosscuts. This method of shooting has produced the best fragmentation in primary blasting.

All drilling is done with air hammer drills, two of which are mounted on tripods for the tunnel and drift work. A driller and a helper are employed on each of these two drills. All the other air hammer drills are operated by one man. Top holes are drilled with churn drills to a depth of 100 ft. The number drilled for a shot varies from two to five. Steel used for air hammers is either 1- or 1¼-in. dressed square point.

Subsequent to the major shots, minor shots are often made by drilling snake holes

into the face at the quarry floor level as deep as 35 ft. These holes are sprung with 40% gelatin. With this type of shooting no top holes are used.

All rock more than 3 ft. in size is broken by secondary shooting. Some of this is done by bulldozing, but most of it is done by drilling shallow holes and shooting with 40% gelatin. All secondary shots are fired with fuse and detonators. Few of the large pieces require more than one hole to break to the desired size.

The coyote and well holes are loaded with bulk dynamite and nitro-starch powders, although some black powder is used. Primary shooting costs average slightly more than one-third of total shooting costs. The labor cost for drilling is about equally divided between primary and secondary shooting.

Loading

Loading of stone or waste is done by two No. 37 caterpillar steam shovels. They are fitted with buckets of 1½ cu. yd. capacity. When in unbalanced production—that is, when it is necessary to waste some of the quarried material—one of the shovels is used to load waste while the other is loading rock. The shovels operate on fuel oil and consume approximately 400 gal. per shift each. Fuel is ordinarily taken on at the middle of the shift when the operations are shut down.

Transportation

All materials are transported in trains of dump cars. The motive power is furnished by three 8-ton gasoline locomotives. Ten cars are in use, six of one type and four of another, all of side-dump design. Tracks are laid with 50-lb. steel and with standard side-throw switches. Fig. 4 shows the track layout. The locomotives use 15 gal. of gasoline per shift and require 5 gal. of lubricating oil every two weeks when operating for one shift. The average haul is 3000 ft. Fig. 7 is an organization diagram which shows the operating personnel.

Drainage

Drainage of the quarry does not offer any particular problem, but in the canyon where the plant is located the run-off during heavy rains is extremely rapid, so that portions of the plant have required walls and light riprap for their protection.

Crushing Plant

A plan and profile of the plant is shown in Fig. 3 and its flow sheet in Fig. 8.

Loaded cars arriving from the quarry are received on a platform above the primary crusher and level with the quarry floor. They are dumped by a specially designed car dump, operated by a 7½-hp. motor, into a steel-lined hopper feeding the primary crusher. The primary crusher is a 48- by 60-in. jaw crusher driven by a 200-hp. motor through a continuous rope drive. This unit is set to crush to 8 in. and discharges into a glory hole or receiving bin, which, together with a recovery tunnel underneath, have been blasted out of the solid rock. From the glory hole the material is fed by a 42-in. pan feeder, driven by a 7½-hp. motor, to a 36-in. belt conveyor, 75 ft. between centers, traveling 180 ft. per min. and

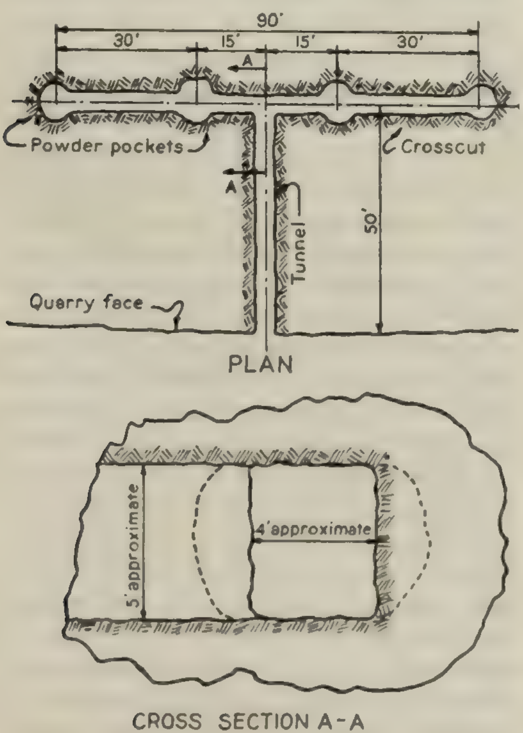

Fig. 5. Typical coyote hole

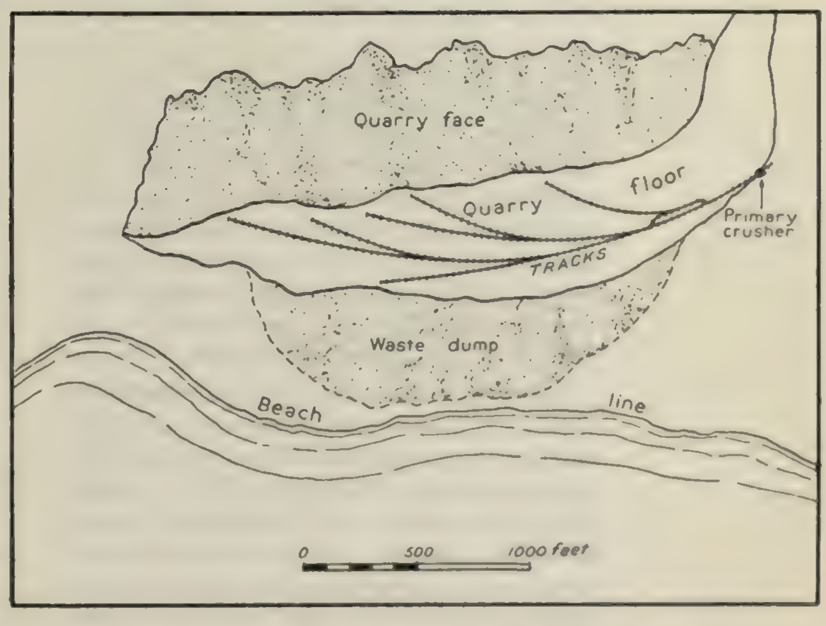

Fig. 4. Plan of quarry, showing track layout

having a capacity of 300 tons an hour. This conveyor is driven by a 7½-hp. back-geared motor and carries the rock to the scalping screen. The original belt, installed when the plant was built, was in operation throughout 1930. The scalping screen, driven through bevel gears, is a 60-in. by 16-ft. trunnion-type trommel set on a slope of 1½ in. per ft., making 12 r.p.m., and fitted with a jacket for one-half of its length. The trommel shell in 1-in. manganese steel has a life of 400,000 tons. The main screen shell has 5-in. holes. The jacket has 1½-in. circular openings. The tonnage passing each screen varies within wide limits, depending on the immediate product required by the market. The material passing the

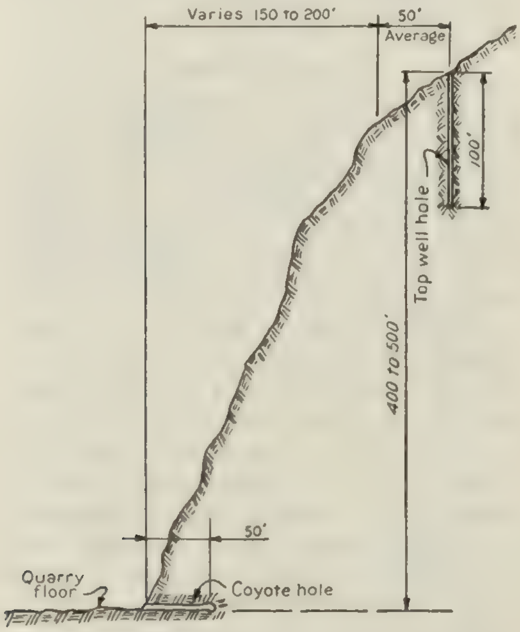

Fig. 6. Cross-section of quarry face, showing position of holes in typical shot

outer jacket is carried on two 24-in. belt conveyors, 70 ft. and 60 ft. in length, to the quarry-fines bin. Both belts are level and travel 180 ft. per min. They were installed in 1930 and are still in use. The material passing the 5-in. circular openings and rejected by the 1½-in. openings is by-passed around the secondary crusher by chutes. (See Fig. 3.) The oversize is fed directly to the secondary crusher.

A No. 16 gyratory set at 2½-in. discharge opening is used as the secondary crushing unit and is driven by a 75-hp. motor. The product of this crusher is fed by gravity over a home-made 4- by 5-ft. flat-bar grizzly with 2-in. spacing. The oversize drops to the by-pass chute and the undersize to a double-decked 4- by 8-ft. vibrating screen mounted on a slope of 2½ in. to the foot. The upper deck is fitted with 1½-in. and the lower with 1-in. square-meshed cloth. No. 8 wire is ordinarily used. The tonnage passing this screen varies within wide limits.

The oversize material is fed through the by-pass chute to a 24-in. conveyor 60 ft. between centers, the first 15 ft. of which is level and the balance of which is on a slope of 12 deg., delivering to the reduction crushers. The speed of this belt is 200 f.p.m. The oversize on the lower deck is discharged into the No. 3 bin below or sent to the next vibrator, as required for grading.

The undersize through the lower deck is delivered to a vibrator screen set on a slope of 2½ in. to the foot and fitted with No. 8 wire cloth to remove dust and excess fines. This wire cloth lasts from 60 to 120 days. The percentage passing this screen varies over wide limits. When handling thoroughly dry material the load is relatively light, as the bulk of the fine material is then removed at the scalper. When wet or damp this screen takes out as much as 30% of its feed. The oversize from this screen is discharged to the No. 3 bin below. The fines are either discharged to the dust bin below to be combined with crusher-run base material, or are fed to a 12-in. belt conveyor 220 ft. between centers traveling 150 f.p.m., which delivers them to a drag washer. This belt is rarely used, however. The washing plant is mounted over a sand bunker to which the washed material is discharged. As there is no good sand on the island, this product has been used to supply local requirements.

The oversize from the vibrator is conveyed by the 24-in. belt conveyor from the by-pass chute to the reduction crushers. These consist of one 4-ft. cone crusher driven by a 100-hp. motor through a short V-belt drive and set at 1½-in. discharge opening, and a 4F gyratory crusher belt-driven by a 60-hp. motor and set at 1½-in. discharge. The proportion of sizes in the finished product is controlled by the proportionate split of the feed between the two units, as well as by their respective settings. Their product is conveyed by two horizontal belt conveyors 190 ft. between centers, extending from under bins below the vibrator screens, one passing under each reduction crusher. The conveyors each travel 190 f.p.m. and are driven by a 7½-hp. motor. They deliver to a bank of four 48-in. by 24-ft. trommels. Between these two belts and drawing oversize from the vibrator is a 6 deg. inclined 24-in. conveyor, 55 ft. between centers, which feeds the reduction crushers. It is driven at 180 f.p.m. by a 5-hp. motor, belt-connected. These screens are operated

in pairs, one pair caring for each of the reduction crushers. Normally, all of the trommels are fitted with a jacket for one-half their length. The jackets have ½-in. round openings. The main shell has 1½-in. round openings. The screens are of the trunnion type, driven by bevel gears at the receiving end. The two outside screens are driven by 15-hp. motors and the middle pair by 20-hp. motors through a belt drive. The main shells are ¾-in. thick and the dust jackets ½-in. The screens are set on a slope of 1½-in. to the foot and rotate at 12 r.p.m. The life of the shells is about 10 months.

The products of these screens fall by gravity into their respective bins and are known in the trade as No. 2, No. 3 and No. 4 rock, the No. 2 rock being the largest size. The oversize from the trommels is discharged by gravity to a 2-ft. cone crusher with discharge opening of 1-in. mounted on the top of the storage bins and belt driven by a 50-hp. motor. The product of this unit is returned to the trommels by an inclined conveyor belt 24 in. wide and 80 ft. between centers, rising 20 ft. in its length and driven by a 10-hp. motor at 180 f.p.m. The operation described is typical; however, larger or intermediate gradings can be made if necessary.

Storing and Shipping

The units just described are mounted on top of the storage bunker, which has four pockets running transversely. At its upper end an additional pocket formed by the sloping surface of the hill and the end of the storage bunker is used for the crusher-run base material. Next to it, in the first pocket, is stored the fines from screening when they are not used for sand as previously described. The next pockets in order hold the three sizes of crushed rock from No. 4 to No. 2. Only a small percentage of the production is utilized on the island, the balance being shipped to the Long Beach distribution terminal. Due to the absence of appreciable local requirements it has not been necessary to provide blending facilities. Since its opening this plant has shipped the three sizes of rock separately. The making of only three sizes was decided on because it reduced the necessary segregation to a minimum and decreased overlap in grading, thus permitting close control of mixes so as to conform to specifications. In many instances the control of mixes has been actuated by the desire to provide low-void aggregates, a field in which this company was the pioneer in this territory. The sizes of rock described are the standard sizes made, although on occasion if the tonnages justify it larger sizes are made.

Passing under the storage bunker is a concrete-lined recovery tunnel. Material being recovered is drawn from the bins to a 24-in. belt conveyor 50 ft. center to center (*A*, Fig. 3), driven at 200 f.p.m. by a 15-hp. motor which discharges to another 24-in. belt conveyor (*B*, Fig. 3) 400 ft. center to center and driven at 200 f.p.m. by a 25-hp. motor which elevates and carries the material to storage under the upper conveyor house. Unusual precautions are taken throughout the plant to remove dust from the crushed rock. In carrying out this object, the upper conveyor-house belt at the top of the elevating incline is carried around two pulleys and discharges its load onto a 4- by 5-ft. vibrating screen set on a slope of 3 in. to the foot and driven by a 1-hp. motor. The purpose of this screen is to remove dust and fines. It is usually fitted with No. 10 wire cloth.

The upper conveyor house is supplied with a mechanical tripper running on rails. The space under the belt is divided into four distinct storage areas. One space each is assigned to the three separate sizes of rock and the fourth to the crusher-run base or quarry waste to which the rock-dust recovered in the crushing plant is added. An additional fixed tripper is located above the vibrating screen to divert, when desired, material to a 1000-ton 4-compartment bunker which cares for local deliveries. This material is handled on a 24-in. by 50-ft. belt, driven at 175 f.p.m. In the conveyor house is a large blower which is used to remove dust from the material being discharged by the movable tripper.

Directly underneath and centered with the conveyor is a concrete-lined tunnel. The material in storage is recovered through gates of the tipping-chute type fitted with rubber skirts in the roof of the tunnel, and is loaded to a 24-in. belt conveyor, 550 ft. center to center, part of which is inclined at 20 deg. (*C*, Fig. 3) running under the full length of the upper conveyor house and through the lower conveyor house. This belt is driven by a 30-hp. motor.

As in the case of the upper belt, at the top of the incline it is carried around two pulleys, discharging its load on a vibrating screen for further removal of dust. This screen is similar to that on the first reclaiming belt. The dust recovered is conveyed to a small dust bunker by a 24-in. by 40-ft. belt driven at 150 f.p.m. by a 5-hp. motor. At present, tests are being made to determine the usability of this dust as a portion of the mix for a clay-products plant located

adjacent to the quarry and operated by the Santa Catalina Island Co. This lower conveyor belt is also fitted with a portable tripper running on rails. The space below is similarly divided into four storage compartments for the separate materials.

Under the lower conveyor house and running its full length is a concrete recovery tunnel. The material is removed from this storage by a 36-in. belt conveyor, 400 ft. center to center and driven at 200 f.p.m. by a 60-hp. motor, which passes over an automatic scale. After passing the scale the conveyor extends across the dock and out to the end of a boom over the water. From this point the material falls by gravity to the waiting barge.

The loading of barges involves a number of features not usually encountered. The location of the loading dock is on the open ocean, on the lee side of the island. While in ordinary weather there are no distinct breakers on the beach, a marked surge is present even on the quietest day. This condition makes it impossible to warp the barges to the dock. Anchors and buoys have been installed so as to hold the barge in position for loading; these are placed at a sufficient distance to give the play necessary to take care of the rise and fall due to swells. The loaded barges are towed to Long Beach by the Wilmington Transportation Co. with Diesel-powered tugs. With the exception of a few days in each year it is possible to load and transport material at all seasons. The barges are usually towed in pairs across the channel, but an additional tug is used to handle the second barge in the harbor. With the exception of one barge which has lines similar to a sailing vessel, all are of the square-end type. In all cases they carry a deck load and vary in capacity from 700 to 1000 tons.

Repair and Drill Sharpening Shops

Minor repairs are cared for at location. Near the superintendent's office is a small shop equipped with a machine lathe, drill press, small tools, and welding outfits. Drills are sharpened in a combination compressor and drill sharpening house which is equipped with an oil-fired furnace and a tool sharpener.

Water Supply

Two sources of water supply have been developed and utilized. Fresh water for use in the steam-shovel boilers is obtained from one of the mines up the canyon, from which it is piped to the quarry level. Due to its character, the water must be sent through a treatment plant, but this procedure has practically eliminated boiler-water troubles. Salt water is obtained from the ocean and is used in the fire-protection system, for wetting down the yards in dry weather, and in the sanitary system. It is supplied at the rate of 15,000 gal. per hour by means of a triplex pump of 7-in. bore, 10-in. stroke, 44 r.p.m., driven by a 40-hp. motor.

Compressor Plant and Air Lines

The compressor house is located on the quarry floor level. It is equipped with one 14x12-in., one 12- by 10-in., and one 10- by 10-in. compressor with individual motor drives totaling 250 hp. The air is delivered at 100 lb. per sq. in. through a system of pipe lines along the entire quarry face. The first 800 ft. is 3-in. pipe, and the next 1600 ft. is 2-in. size. Bleeders are placed every 600 ft. and tees are put in the line at convenient points for 1½-in. feeders to the quarry. A small air line is carried through the crushing plant for use in blowing out accumulated dust when cleaning up.

The compressors used are as follows:

One 14- by 12-in., 100 hp., 268 r.p.m.
One 12- by 10-in., 75 hp., 245 r.p.m.
One 10- by 10-in., 75 hp., 268 r.p.m.

Power Supply

With the exception of the steam shovels and locomotives, all power used in this operation is furnished by electricity. This is obtained from a central generating station in Avalon operated by the Santa Catalina Island Co. All the power is generated by Diesel-driven dynamos. It is transported at 11,000 volts about 1½ miles over a high-tension power line to a group of three 250-kilowatt transformers on the quarry level, where it is stepped down to 440 volts.

Fuel

The steam shovels are fired with 14-gravity fuel oil. This is transported in bulk from the mainland. Between the upper and lower conveyor houses is a fuel tank, from which the fuel oil is pumped to the quarry level by a 3- by 6-in., duplex, 2½-in. suction, 2-in. discharge pump driven by a 5-hp. motor at 39 r.p.m.

General Operation

The policy of the company has been to produce the best possible materials which can be profitably marketed. Since opening this plant, operations have altered only in the direction of perfecting practice. As all processes are dry, it is necessary to use every practical means to remove dust from the product. In the early period of the plant's

operation material now marketed as crusher-run base was almost entirely wasted, but research into the possibilities of marketing it, followed by an intensive educational campaign setting forth its excellent physical properties, has resulted in considerable sales. This crusher-run base consists essentially of a well-graded crushed rock mixture with sufficient finely disseminated clay to bind and thoroughly cement it. Below is given an average analysis of the material, together with the clay content and its distribution.

AVERAGE SCREEN ANALYSIS AND CLAY CONTENT OF CRUSHER-RUN BASE

Size Passing:	Retained on:	Amount, per cent.	Clay content, per cent. of total	Clay, per cent. of fraction
1½-in. circle,	½-in.	12.7	0.3	2.4
½-in. circle,	¼-in.	19.1	0.8	4.2
¼-in. circle,	10-mesh	25.0	0.7	2.8
10-mesh,	20-mesh	14.4	1.0	6.9
20-mesh,	30-mesh	4.7	0.5	10.6
30-mesh,	40-mesh	4.6	0.6	13.0
40-mesh,	50-mesh	3.2	0.4	12.5
50-mesh,	80-mesh	3.7	0.6	16.2
80-mesh,	100-mesh	2.2	0.3	13.6
100-mesh,	200-mesh	3.6	0.8	22.2
200-mesh		6.8	3.8	56.0
Total		100.0	9.8	......

The opening of a market for this product, which has consistently grown each year, has had a double advantage. Much less of the material shot down is wasted and it is not so necessary to make a sharp cut between sound rock and waste in the upper portions of the crushing plant. The sale of crusher-run base has economically justified more stages of dust removal and has thus materially improved the quality of the crushed rock.

The grades of rock produced have varied somewhat. During the past two years most of the production has been in the three sizes of which screen analyses follow:

No. 2 ROCK

Passing:	Retained on:	Per cent.
2½-in. circle	2 -in. circle	63.8
2 -in. circle	1½-in. circle	27.1
1½-in. circle	1 -in. circle	9.1
		100.0

No. 3 ROCK

Passing:	Retained on:	Per cent.
1½-in. circle	1-in. circle	40.8
1 -in. circle	¾-in. circle	38.8
¾-in. circle	½-in. circle	15.6
½-in. circle	¼-in. circle	4.8
		100.0

No. 4 ROCK

Passing:	Retained on:	Per cent.
½-in. circle	¼-in. circle	49.7
¼-in. circle	10-mesh	50.3
		100.0

Mixes with either 2½-in. or 1½-in. maximum size are consistently delivered with voids below 38%.

PER CENT. EXTRACTION

	Per cent.
Crushed rock delivered to the mainland	48.9
Material delivered locally	4.2
Quarry fines delivered to the mainland	23.3
Waste	23.6
	100.0
Recovery	76.4

TABLE 1. LABOR AND WAGE SCHEDULE

Classification	Department	No.	Rate per hour	Monthly salary	Hours per shift	Days per year
General superintendent	Quarry and mill	1	....	$350	....	......
Quarry foreman	Quarry	1	....	300	....	......
Blacksmith	Quarry	1	$0.85	....	8	300
Blacksmith's helper	Quarry	1	.55	....	8	300
Compressor operator	Quarry	1	.75	....	8	300
Steam shovel operator	Quarry	2	....	250	....	......
Steam shovel firemen	Quarry	2	.75	....	9	300
Steam shovel pitmen	Quarry	2	.60	....	8	300
Hoistman	Quarry	1	.60	....	8	300
Locomotive drivers	Quarry	3	.60	....	8	300
Trackman	Quarry	1	.75	....	8	325
Powderman	Quarry	1	.75	....	8	300
Drillers	Quarry	8	.60	....	8	300
Muckers	Quarry	4	.50	....	8	300
Master mechanic	Mill	1	....	250	....	......
Handyman	Mill	1	.50	....	varies	varies
Mill labor boss[1]	Mill	1	.70	....	8	300
Mill labor boss[2]	Mill	2	.60	....	8	300
Mill labor boss[3]	Mill	10	.50	....	8	300
Electrician	Mill	1	.85	....	varies	varies
Labor boss	Storing and shipping	1	....	200	....	......
Labor	Storing and shipping	3	....	150	....	......
Labor, extra	All departments	....	.50	....	....	......

[1] In charge of mill operations on the second shift.
[2] One on each shift.
[3] Six on day shift and four on night shift.

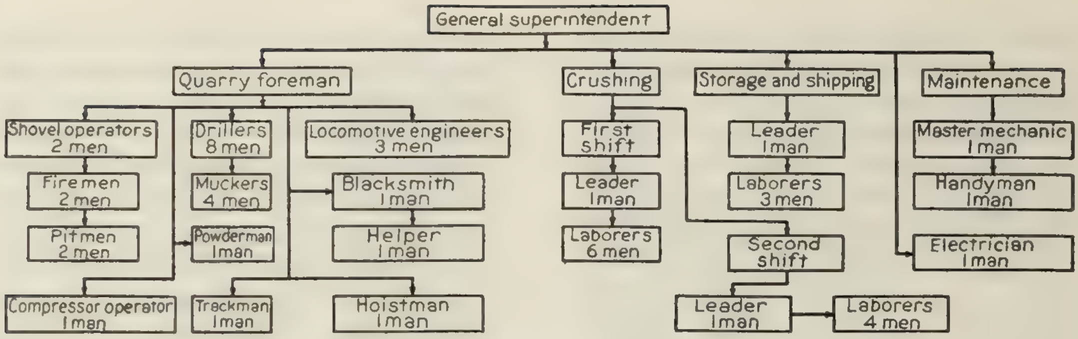

Fig. 7. Organization diagram

Fig. 9 shows the quantity and size of material shipped by months during 1930.

During 1930 the quarry floor crew worked for only one shift, during which they were able to produce and deliver sufficient material for a 2-shift mill operation. The primary crusher can easily care for this load. During the second shift, material is withdrawn from the glory hole and secondary storages and passed through the recrushing units.

Safety Methods

The quarry and mill are subject to three separate sources of safety inspection. The State of California, through the Industrial Accident Commission, maintains an effective and rigid inspection and control of operations of this character; the compensation insurance carrier also makes periodic inspections; but probably the most effective safeguard is the work of the company's own safety engineer. All the activities of the Santa Catalina Island Co. and Graham Bros. (Inc.) on Santa Catalina Island are under the supervision of the same safety engineer. He has worked out a system for

TABLE 3. SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Santa Catalina Island Quarry. Period January 1, 1930, to December 31, 1930.

Material shipped during period: 409,283 tons.

Labor (man hours per ton):	
Drilling	0.072
Blasting	.006
Loading	.069
Haulage	.034
Miscellaneous	.003
Supervision	.007
Total quarry labor	0.191
Labor, per cent. of total cost	57.5
Power and supplies:	
Explosives (pounds per ton)	0.51
Total power (kw. h. per ton)	4.85
Air compressors	1.23
Pumping	0.11
Crushing	2.17
Screening	0.43
Conveying	0.30
Storage	0.51
Shops	0.06
Lighting	0.04
Fuel (barrels per ton)	0.014
Other supplies, per cent. of total	51.0

Supplies and power are 42.1% of direct cost and 27.2% of total operating cost.

This analysis is based on tonnage shipped.

TABLE 2. SUMMARY OF COSTS

Santa Catalina Island Quarry. Period January 1, 1930, to December 31, 1930.
Total material shipped: 409,283 tons.

	Cost per dry ton of stone shipped					
	Labor	Power	Fuel	Explosives	Supplies	Total
Drilling: Primary	$0.0206	$0.0135	....	....	....	$0.0559
Secondary	.0218					
Blasting: Primary	.0045	....	....	$0.0193	....	.0545
Secondary				.0307		
Loading	.0353	....	$0.0074	....	....	.0427
Hauling	.0184	....	.0035	....	....	.0219
Undistributed quarry[1]	.0044	....	....	....	$0.0063	.0107
Crushing	.0688	.0239	....	....	.0275	.1282
Screening		.0047	....	....		
Conveying		.0033	....	....		
Storage	.0308	.0056	....	....	.0095	.0459
Loading						
Undistributed mill[2]	.0253	.0023	....	....	.0202	.0478
Direct administration	.0147	....	....	....	....	.0147
Total operating direct	$0.2446	$0.0533	$0.0109	$0.0500	$0.0635	$0.4223
Depreciation	....	....	....	....	....	.1324
Royalty	....	....	....	....	....	.0798
Compensation insurance	....	....	....	....	....	.0195
Total operating cost	....	....	....	....	....	$0.6540

[1] Includes foreman, and repairs to shovels, locomotives, cars, etc.
[2] Labor cleaning plant, power for lights, water system, etc.

grading based on cleanliness, efficiency, and effort to improve. Each month a report is made showing the relative standing of the various activities. Any foreman or superintendent who does not maintain his rating at 95% or better is discharged. An operation is considered as satisfactory if it maintains an average standing of 95% or better. The quarry and mill had a general average for 1930 of 98.6%. A 100% rating is given only when conditions are exceptionally fine and is accompanied by an honorable mention in the monthly report. This operation had such honorable mention three times during the year. One man each at the quarry and mill is made reponsible for reporting any unsafe condition.

Administrative Organization

The general superintendent is in complete charge of the quarry, mill, and shipping.

TABLE 4. DETAILED AVERAGE SHOVEL COSTS, DIRECT OPERATION
(Average of two identical shovels)

Santa Catalina Island Quarry. Period from January 1, 1930, to December 31, 1930.
No. 37 Marion steam shovel, caterpillar. 1½ cu. yd. bucket.

Material handled	Tons
Rock and quarry fines	204,640
Waste	63,200
Total	267,840

	Rock and quarry fines		Waste		Total
	Amount	Cost per ton	Amount	Cost per ton	Cost per ton
Engineer	$2,292.00	$0.0112	$ 718.00	$0.0112	$0.0112
Fireman	1,411.87	.0068	445.11	.0068	.0069
Pitman	1,136.83	.0056	351.47	.0056	.0056
Other operating labor	663.09	.0033	204.53	.0033	.0033
Total operating labor	$5,503.79	$0.0269	$1,719.11	$0.0269	$0.0270
Fuel and lubricants	1,156.96	.0057	357.39	.0057	.0057
Total	$6,660.75	$0.0326	$2,076.50	$0.0328	$0.0326

Note—The total figure of $0.0326 per ton on total tonnage handled would become $0.0427 per ton shipped, as in Table 2.

Repairs and other miscellaneous charges on all equipment used in the quarry operation amount to $0.0063 per ton shipped or $0.0048 per ton handled. This account is undistributed and therefore the portion chargeable to direct shovel operation could not be determined and is not included in the above schedule.

TABLE 5. DETAILED SUMMARY OF COSTS OF DIRECT OPERATION

Santa Catalina Island Quarry. Period from January 1, 1930, to December 31, 1930.

	Rock and quarry fines, 409,283 tons		Waste, 126,400 tons		Total, 535,683 tons
	Amount	Cost per ton	Amount	Cost per ton	Cost per ton
Total all shovels (Table 3)	$13,321.50	$0.0325	$ 4,153.00	$0.0329	$0.0326
Drilling:					
Operating labor	14,812.87	.0362	2,540.73	.0201	.0324
Power	4,717.20	.0115	808.12	.0064	.0103
Total drilling	$19,530.07	$0.0477	$ 3,348.85	$0.0265	$0.0427
Blasting:					
Labor	1,675.20	.0041	166.57	.0013	.0034
Explosives	18,597.68	.0454	1,864.32	.0148	.0382
Total blasting	$20,272.88	$0.0495	$ 2,030.89	$0.0161	$0.0416
Hauling:					
Locomotive drivers	3,395.24	.0082	1,045.20	.0082	.0082
Locomotive fuel	1,094.67	.0027	337.82	.0027	.0027
Track labor	1,415.13	.0035	437.13	.0035	.0035
Dump labor	943.20	.0023	291.42	.0023	.0023
Total hauling	$ 6,848.24	$0.0167	$ 2,111.57	$0.0167	$0.0167
General charges[1]	4,205.68	.0103	165.60	.0013	.0082
Grand total	$64,178.37	$0.1567	$11,809.91	$0.0935	$0.1418

[1] Includes cost of direct administration, repairs, and other miscellaneous costs.
Note—the above total cost per ton referred to tonnage shipped becomes $0.1857.

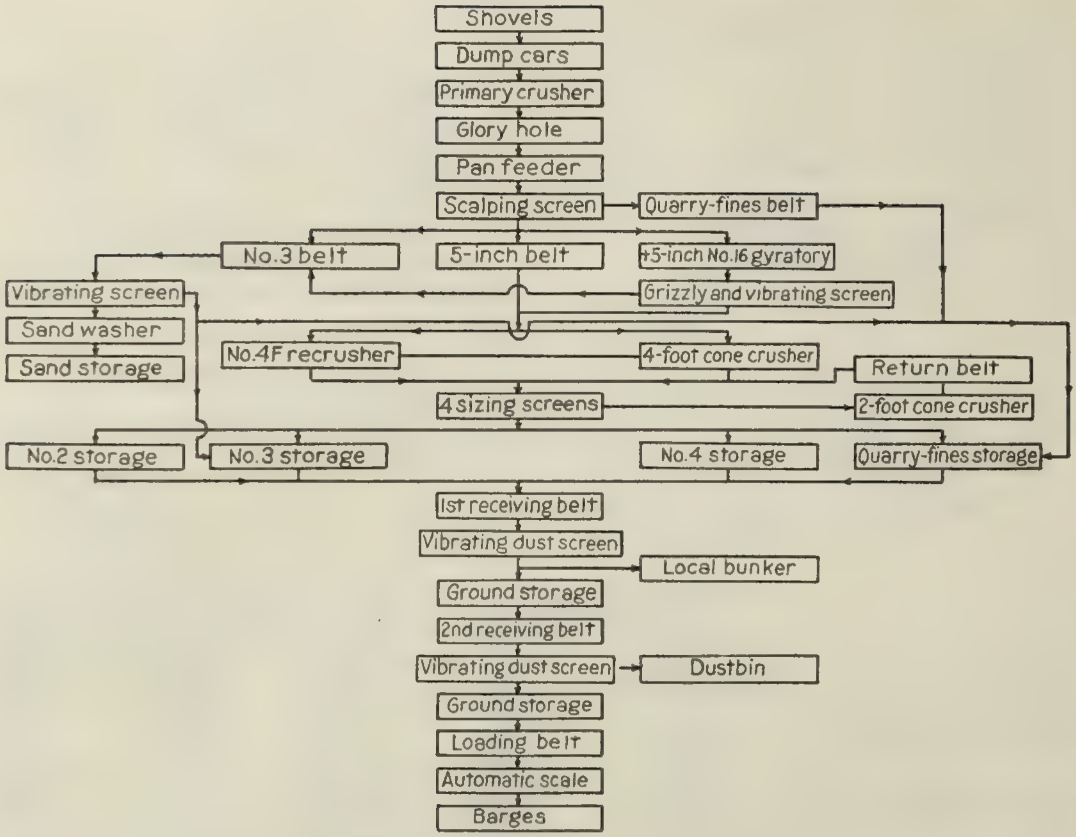

Fig. 8. Flow sheet

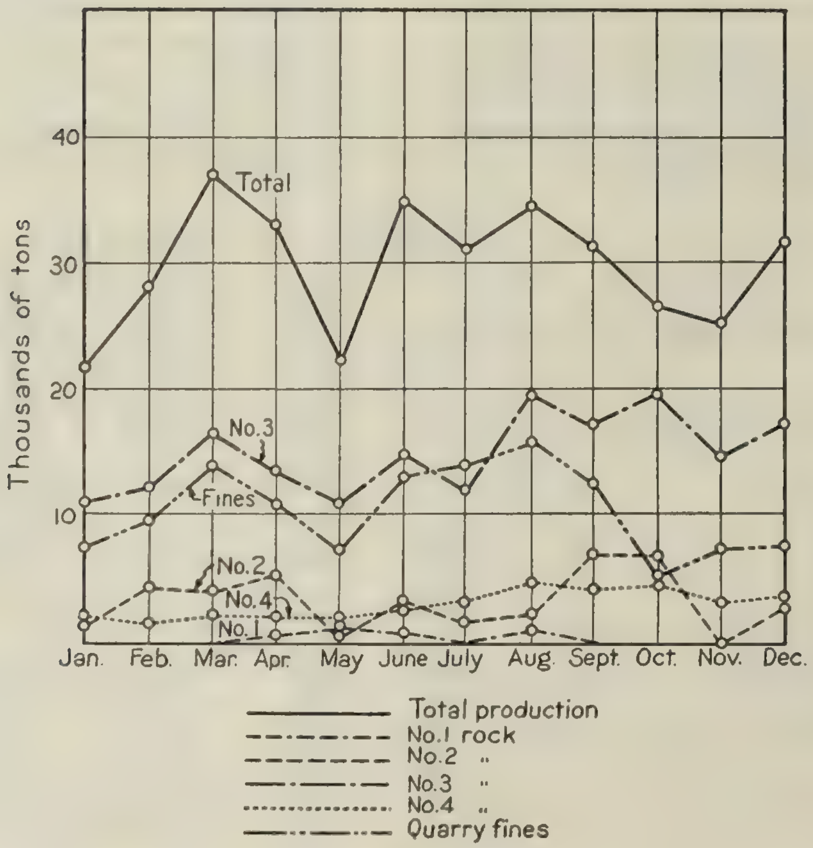

Fig. 9. Production chart for 1930. Material shipped to mainland

On the quarry floor and in incidental activities such as those of the blacksmith shop, water-treatment plant, compressor house, handling and storing of explosives, and transportation and dumping of material, the general superintendent is ably assisted by the quarry foreman. In the mill each stage of crushing is in charge of one of the workers. There is a foreman in the storage and shipping department. All maintenance and repair work is supervised by the master mechanic or electrician in their respective fields. Fig. 7 is an organization chart which shows the personnel. This chart includes the men employed on the second shift in the mill.

TABLE 6. LIST OF MOTORS, SANTA CATALINA ISLAND QUARRY

1—100-hp. sleeve-bearing, driving 14- by 12-in. compressor. Short-belt drive.
1— 75-hp. sleeve-bearing, driving 12- by 10-in. compressor. Short-belt drive.
1— 75-hp. sleeve-bearing, driving 10- by 10-in. compressor. Short-belt drive.
1— 2-hp. electric, ball-bearing, emery wheels, in blacksmith shop.
1— 5-hp. ball-bearing, used on top of quarry face to hoist steel for well drills.
1— 10-hp. ball-bearing. Car-dump hoist.
1— 5-hp. sleeve-bearing fuel pump.
1—200-hp. ball-bearing, driving 48- by 60-inch. jaw crusher. Belt drive to countershaft, rope drive countershaft to crusher.
1— 75-hp. ball-bearing, driving No. 16 primary breaker belt drive.
1— 15-hp. ball-bearing, driving scalping screen. Belt drives motor to countershaft, countershaft to screen.
1— 10-hp. driving double-decked shaker screen. V-belt drive.
1—7.5-hp. ball-bearing, driving pan feeder from glory hole.
1—7.5-hp. ball-bearing, driving 36-in. conveyor from pan feeder.
1— 5-hp. ball-bearing, driving 24-in. conveyor from shaker screen.
1— 5-hp. ball-bearing, for dust collector driving No. 1 fan.
1— 5-hp. sleeve-bearing, driving quarry fines belt from scalping screen.
1— 3-hp. sleeve-bearing, driving machine-shop equipment.
1—100-hp. ball-bearing, driving No. 4 cone crusher. Short drive.
1— 60-hp. ball-bearing, driving No. 4 recrusher. Belt drive.
1— 50-hp. ball-bearing, driving 2-ft. cone crusher. Belt drive.
1— 15-hp. ball-bearing, driving west screen sizing.
1— 15-hp. ball-bearing, driving east screen sizing.
1— 20-hp. ball-bearing, driving middle east sizing.
1— 20-hp. ball-bearing, driving middle west sizing.
2—7.5-hp. ball-bearing, driving two belts to sizing screens.
1—7.5-hp. ball-bearing, driving conveyor to sand washer.
1— 10-hp. ball-bearing, driving No. 12 fan.
1— 5-hp. ball-bearing, driving belt feeding No. 4 recrusher and cone crusher.
1— 5-hp. oversize screens to 2-ft. cone crusher.
1— 25-hp. sleeve-bearing, driving long belt, upper storage house.
1— 15-hp. ball-bearing, pick-up belt, feeding upper storage-house belt.
1— 5-hp. sleeve-bearing, driving belt to local demand bunkers.
1— 1-hp. sleeve-bearing, driving vibrating screen.
1— 10-hp. ball-bearing, driving fan. Direct drive.
1— 30-hp. ball-bearing. Lower storage house.
1— 5-hp. ball-bearing. Dust belt from vibrating screen.
1— 1-hp. sleeve-bearing, vibrating screen.
1— 60-hp. ball-bearing, driving loading belt.
1— 40-hp. sleeve-bearing, salt water pump.
1— 10-hp. ball-bearing. Barge winch.

I. C. 6610

FEBRUARY, 1933

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

METHOD AND COST OF QUARRYING LIMESTONE AT THE PLANT OF THE CALAVERAS CEMENT CO., SAN ANDREAS, CALIF.

BY

ROBERT H. TOWNSEND

I.C. 6610,
February, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHOD AND COST OF QUARRYING LIMESTONE AT THE PLANT OF THE CALAVERAS CEMENT CO., SAN ANDREAS, CALIF.[1]

By Robert H. Townsend[2]

INTRODUCTION

The following paper discusses the mining, transportation, and crushing methods, with costs, at the San Andreas plant of the Calaveras Cement Co. It is one of a series, prepared for and published by the U. S. Bureau of Mines, describing methods used in cement-rock quarries throughout the country.

These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total costs. It is recognized that publication of total costs mightin some instances cause embarrassment to individual producers as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of the methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

ACKNOWLEDGMENT

Acknowledgment is made to Dave Wheeler, engineer of the company, for the drawings and man-hour data; to Harold Dunton, chief-chemist, for analyses; and to E. E. Halley, chief-clerk, for the cost data in this paper. Information regarding the geology of the district was obtained from the Jackson Folio of the United States Geological Survey.

1/ The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6610."

2/ One of the consulting engineers, U. S. Bureau of Mines, and superintendent, Calaveras Cement Co.

HISTORY

Through the efforts of William Macnider the possibilities of the limestone deposits at San Andreas were brought to the attention of W. W. Mein and Stuart L. Rawlings, mining engineers, in the summer of 1923. An examination of the properties proved the limestone to be favorable for cement making. Options were obtained on approximately 1,500 acres, and plans for a plant of 1,200,000 barrels yearly capacity at Kentucky House were submitted and approved. Construction was started in 1925, and the first cement was shipped in June, 1926.

It was decided to first work No. 1 hill, approximately 1 mile north of the plant, using 18 and 20 ton gasoline locomotives with 18-ton cars for the transportation of rock to the plant.

GEOLOGY

The Calaveras formation, in which are found the limestone lenses and shale that supply the plant, occurs as an isolated area at this point east of the Bear Mountain and has a length of about 12 miles and an average width of 1/2 mile. It is composed of black clay-slate, quartzite, and limestone lenses. The lens occurring at Kentucky House is approximately 1-1/2 miles in length. Kentucky House was formerly the name of a ranch house located on the present plant site and is now the name of the railroad station serving the cement plant and terminal of the Southern Pacific line from Lodi, Calif. To the west of this narrow area are the black slates of the Mariposa beds. The beds are indicated as being of the Carboniferous age, although this is not certain.

Amphibolite and amphibole-talc rocks have been outlined within the central mass in small areas, and other areas of hornblende and mica schists occur. The entire series is much altered and when associated with schists representing original sediments, the differentiation of the schists in the field is difficult. The shale used at the plant is supplied by the same quarry from which the limestone is obtained.

The general trend of the strata in this vicinity is about north 20° west. The dip, which is steeply to the east, is 72° at the quarry.

The limestone varies from dark blue to nearly white, and is irregular in composition at different points along the lens. At the northerly end the beds, where exposed, apparently pinch out and show a much lower lime content and decidedly more silica than at the southerly end. Analyses of high and low grade rock are as follows:

	Limestone, per cent	
	High-grade	Low-grade
Silica	2.30	37.05
Iron and aluminum oxides	1.39	16.73
Calcium carbonate	95.57	32.85
Magnesium carbonate	.93	3.63

These grades are used in the proportion required to make a mix.

METHODS OF PROSPECTING AND EXPLORATION

Prospecting of the properties was originally done by William Macnider and the Riverside Portland Cement Co. No costs on the original prospecting and testing can be supplied, as this work was completed before the present management assumed control of the properties and it does not have the cost records. Surface trenching was carried out at intervals of 200 feet across the width and at right angles to the strike of the deposit. Some core-drill holes were sunk, and the cores were checked against trench samples. A tunnel 700 feet long was driven to crosscut the highest-grade limestone 100 feet below the surface exposures. This crosscut was carefully sampled and checked against samples from surface trenches in the immediate vicinity.

The sample trenches were 2 feet wide and their depth was dependent on the thickness of surface shale, as all trenches were carried to solid rock. Samples were taken by chipping small pieces from the rock at definite intervals, in both the tunnel and trenches.

A sufficient volume of cement rock was proved at Kentucky House in conjunction with a high-grade limestone deposit 6 miles east at Old Gulch to warrant the installation of a plant.

Samples from 14 well drill holes subsequently sunk to a depth of 185 feet on the best lens showed that plenty of material was available for a third bench in quarrying. Samples were taken at every 10 feet, the analyses of which follow.

Analyses of material from 14 well drill holes, per cent

Hole No.	Silica	Iron and aluminum oxides	Calcium carbonate	Magnesium carbonate
4	3.09	1.69	95.5	.15
5	6.75	2.70	89.73	1.05
6	1.43	1.35	97.28	.52
7	7.09	3.08	88.94	1.26
8	1.13	1.08	96.10	1.84
9	1.79	.86	96.70	1.16
10	1.59	.79	96.28	1.48
11	2.68	2.13	[illegible]	1.78
12	4.21	1.54	92.77	2.63
13	5.84	1.50	91.62	.75
14	1.77	.57	95.89	1.43
15	4.21	1.28	93.46	.71
16	10.06	4.31	83.08	1.97
19	2.00	1.55	94.23	1.83

METHOD OF SAMPLING AND ESTIMATION OF TONNAGE AND VALUES

At the north end of the quarry and in some parts of the south end the limestone is interspersed with streaks of shale. These shale streaks are mined with the limestone.

The quarry face at the south end, in high-grade limestone and nearest the plant (see fig. 1), is approximately 900 feet long. The low-grade rock from the north end is obtained from a through-cut which will eventually establish a quarry face. These faces are mined independently; the stone is brought to the plant and crushed, run over a grizzly, and stored separately. Rock from the different piles is mixed by the cranemen, who take a certain number of buckets of one kind to so many of the other, as specified by the chemist, and dump them into the bins feeding the compeb mills.

The floor of the top bench at the quarry is 1,000 feet above sea level, and the bench face is from 50 to 60 feet high. There is no overburden to dispose of, as the surface shale is mined with the rock and utilized at the plant. An attempt is made to keep a straight quarry face, but in some cuts the rock at one end has been lower in grade than at the other, and this has at times necessitated a slight bow in the face.

Reserve tonnages are computed by determining the volume of stone between successive rows of well drill holes. This tonnage is checked by mapping the back-break after each shot and figuring the average distance from the back-break of the preceding shot. The unit weight of stone in place is figured at 160 pounds per cubic foot, or $12\frac{1}{2}$ cubic feet to 1 ton.

Drainage up to the present has presented no problem, as both the 1,000-foot and 950-foot benches drain to the creek.

Figure 1 shows the track layout at the quarry.

MINING METHODS

Up to the present, benches 50 feet high have been taken. Parts of the face on the top bench have reached a height of 75 feet, but no difficulty has been encountered with slides. It is probable that the third bench, when it is necessary to open it, will have a face of 70 to 90 feet.

The quarry face at the south end of Hill No. 1 was opened on the west side of the hill, normal to the west slope, in order to attain a maximum length of quarry face in the high-grade limestone. A large percentage of the rock east of the crest of the hill shows a low-lime content, making it impracticable to open the face normal to the strike. The present quarry face is roughly parallel to the strike and is being advanced to the east. Although the dip, as previously mentioned, is 72° east, no difficulty is experienced from overhang after shooting, due to the fact that the limestone lens is massive, and the stratification is not clearly defined. The average back-break on the 50-foot face is 18 feet, giving a safe working place during the clean-up.

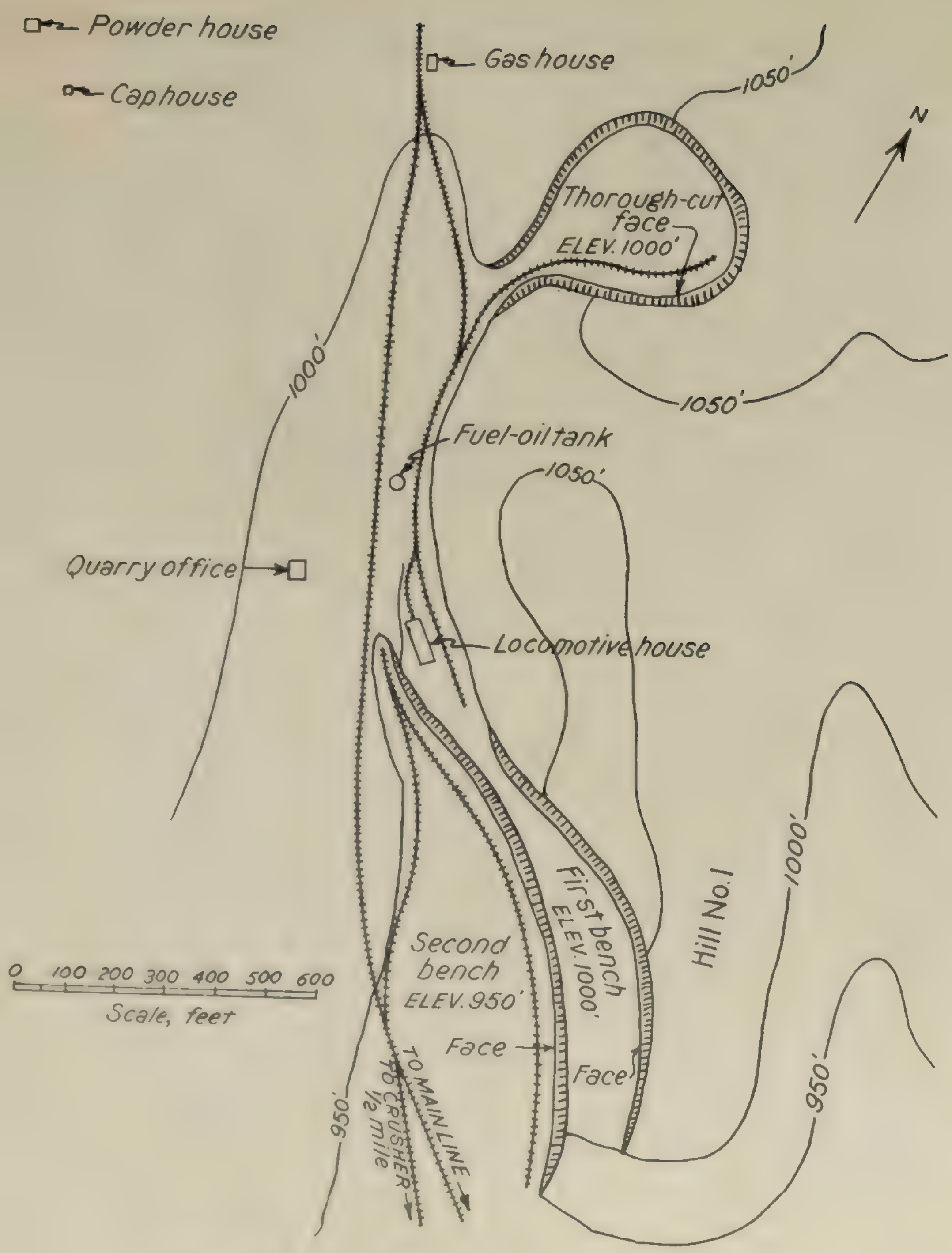

Figure 1.- Quarry and track layout

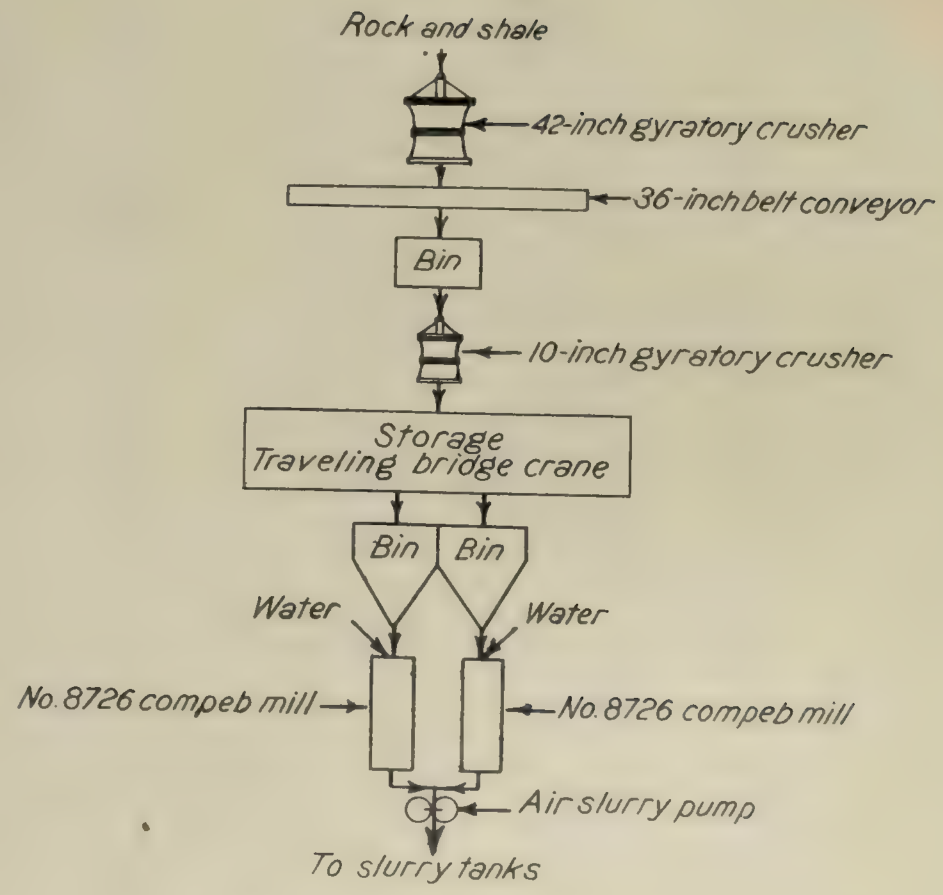

Figure 2.- Flow sheet of crushing department

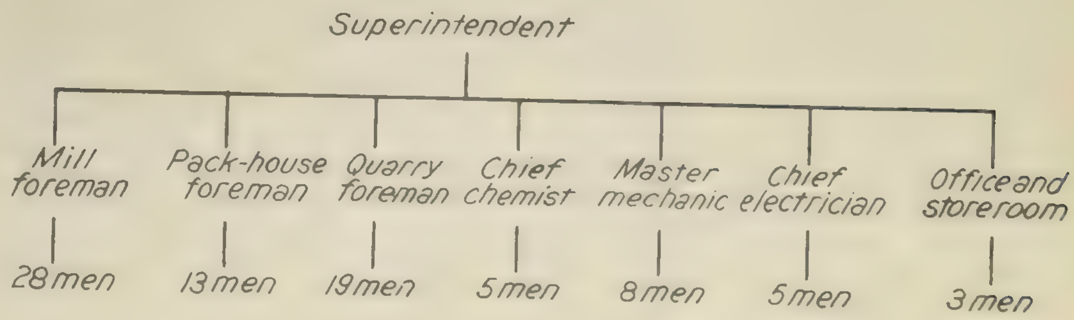

Figure 3.- Organization chart

At present the thorough-cut face is being worked transverse to the strike.

At full capacity the quarry delivers 1,000 tons of rock and shale a day. This is handled in an 8-hour shift by a 50-B alternating-current shovel of 2 yards dipper capacity and a 60-C converted railroad shovel of $2\frac{1}{2}$ yards dipper capacity. At half production one shovel is worked at a time, the crew being transferred from one face to the other.

Eighty thousand to 120,000 tons of rock are shot down at a time. Due to crusher delay from sticky shale, 20,000 to 30,000 tons of rock and shale are usually carried in storage at the plant during the rainy season. The shovels are worked in limestone when heavy rains are prevalent.

DRILLING AND BLASTING

Drilling is done with a 6-inch 50 B-E well drill, driven by a 15-hp., 440-volt, 1,200-r.p.m. induction motor.

Holes are drilled 18 feet apart along the quarry face and about 35 feet between rows. From 30 to 50 holes are drilled in a row and shot together.

The drill bits are of steel, 6 inches in diameter, with a cutting edge of 7 inches beveled to 45°. One driller and helper work 8 hours a day and drill from 20 to 24 feet in solid limestone. The driller is paid $200 a month and works 27 days. The helper receives $62\frac{1}{2}$ cents an hour. Drilling cost per foot, including repairs, overhead, and depreciation, varies from $0.85 to $1.40, depending upon the character of the rock. From 42 to 45 tons are broken per foot of hole and the weight of powder per ton of rock broken ranges from 0.25 to 0.33 pound. Holes are drilled to a depth 5 feet below the floor of the bench. Holes on the 950-foot bench are all 55 feet deep.

BLASTING

The amount of powder per hole will vary, dependent upon whether solid limestone, creviced limestone, or limestone and shale is cut. The stone is blocky and breaks big. An average row of holes along the face will use about 17 to 18 50-pound boxes of 40 per cent free-running powder, and a 60-pound box of 60 per cent gelatin dynamite, which is placed in the bottom of the hole. Holes are sprung with from 50 to 150 pounds of 60 per cent gelatin, the quantity used depending on the nature of the rock. The holes are drilled and sprung several weeks prior to loading. They are then loaded with about 12 boxes of bag powder, stemmed, and the balance of the powder is loaded in the barrel at certain depths up to within 25 feet of the top. In this manner the bottom is broken up and kicked out, and the upper part of the rock is also shattered yet not thrown out too far.

Cordeau is used in all drill-hole blasting and is exploded by a No. 8 detonator.

Dry, screened quarry shale is used for stemming. Tamping is done with a wooden dolly on a rope.

Because of the large blocks sometimes resulting from primary blasting, the amount of secondary shooting required is appreciable. Secondary blasting is accomplished by drilling with jackhammers and shooting with 40 per cent gelatin to primary-crusher size. Powder used for secondary shooting will average about 0.068 pound per ton of rock, giving an average ratio of 4.4 to 1 between the quantities of explosive used in primary and secondary shooting.

LOADING STONE

One No. 50-B alternating-current shovel and one No. 60-C, both mounted on caterpillar tractors, load all the stone. The power plant on the 50-B shovel consists of one 80-hp. hoist motor, two 33-hp. motors for boom and swing, one 1-hp. trip motor, and a 2-hp. compressor motor. The 60-C shovel has one 115-hp. hoist motor, two 50-hp. boom and swing motors, one 10-hp. steering motor, and a 3-hp. compressor motor. Power is delivered to the quarry shovels at 2,200 volts. The power line is carried by overhead construction from the plant to a central quarry distributing point located about halfway between the thorough-cut and south quarry face. Power is carried through a surface conduit from the central distributing point to a safe working distance from the face, and thence through flexible rubber-covered cable directly to the shovel transformers, where it is reduced to 440 volts. The 50-B shovel is operated by one man and the 60-C shovel by two men.

TRANSPORTATION

Transportation from the quarry to the plant is effected by means of one 13-ton and one 20-ton gasoline locomotive, and a 25-ton Diesel locomotive is used as an auxiliary haulage unit, as only two trains are operated at one time.

Twelve steel solid-box rock cars (see fig. 4) with a capacity of 18 tons each are used. Both ends of the cars are supplied with swiveled couplings.

Two and three car trains are run directly from the shovels to the crusher; no switching engine is used. The cars are backed into the tipple by the locomotive, dumped, and returned to the shovel. The crusher is located at an elevation of 910 feet, and there are no adverse grades for loaded trains from the quarry to the crusher. Empty trains are hauled against a grade averaging 1.6 per cent from the crusher to the second bench and 2.5 per cent from second bench to the thorough-cut. The haul from the second bench to the crusher is 3,000 feet, and from the thorough-cut to the crusher is 1 mile.

The quarry track is all standard gage 4 feet $8\frac{1}{2}$ inches, and 65-pound rail is used.

In the development of the quarry it was deemed advisable to use switchbacks for track leads to both faces, in order to secure better alignment and lighter grades on the quarry tracks. As the time required for a shovel to load a train is approximately the same as that taken for a train to run to the crusher, discharge its load and return to quarry, these switchbacks provide convenient passing tracks, thus obviating the necessity of constructing passing switches.

CRUSHING PLANT

The 18-ton cars from the quarry are run onto the tipple (see fig. 5) above a 42-inch gyratory crusher and dumped by compressed air. One man operates the tipple and the crusher, which is belt driven by a 200-hp., 440-volt, 765-r.p.m. motor.

The 6-inch rock discharge from the primary crusher is carried by a 36-inch conveyor belt 206 feet between centers up an incline of 15° at a speed of 300 f.p.m. to a 10-inch secondary crusher with the same type of belt driven by a 100-hp., 440-volt, 865-r.p.m. motor. The conveyor belt is 6 ply with 1/4-inch rubber on top, 1/16-inch rubber on the bottom, and a total thickness of 11/16 inch. The belt discharge passes over a $4\frac{1}{2}$ by 7 foot grizzly set on a slope of 40° and having $1\frac{3}{4}$-inch openings, the fines from which are sent through a chute to the rock storage. The grizzly bars are manganese steel, 3 inches wide, set on edge, and taper in thickness from 1 inch on top to 3/4 inch on the bottom. The oversize is fed to the secondary crusher, which is set at $1\frac{3}{4}$-inch opening and discharges its product to a separate pile in the rock storage.

The shale from the quarry is more easily crushed than the stone, and practically all shale will fall through the grizzly after passing the primary crusher. Hence the grizzly really acts as a classifier in separating the shale from the rock, and both are chuted to separate places in the rock storage.

ROCK STORAGE

The rock storage is simply a storage space 80 feet wide and 185 feet long, in three sections or bays divided by walls 52 feet high and having a total capacity of 25,000 tons. One end is reserved for shale, the center bay for low-lime rock, and the other end for high-lime rock.

The secondary crusher and grizzly chutes discharge into one side of the storage. An 8-ton electric traveling crane transfers all raw material from the crusher and grizzly discharge to piles within the storage area and from there to the raw-material bins on the opposite side of the storage as needed. The crane is equipped with a 3-cubic yard bucket and is operated by one 10-hp. carriage motor, one trolley, one hoist, and one closing-line motor, all of 50-hp. capacity. All motors are of variable speed and operate at 440 volts.

GRINDING

Grinding is accomplished in two 2-compartment compeb mills. They are 8 and 7 by 26 feet. A 35-inch table feeder, driven by clutch, chain, and sprocket from the mill, feeds the raw material, 1-1/2 inches in size and less, to the mills. Four and one-half inch grinding media are used in the preliminary end and 1-inch steel balls for the finish grind. The mills are driven at 21 r.p.m. by two 500-hp., 2,200-volt, 180-r.p.m. synchronous motors. Consumption of grinding media is 1.1 pounds per ton of material ground. Ribbed liner plate consumption amounts to 0.10 pound per ton ground. Ninety per cent of the slurry discharged from the mills will pass a 200-mesh screen, and has a moisture content of 34 per cent. The mills each grind an average of 22.6 tons per hour of the harder rock from the north end of the quarry. On pure limestone and shale this has been raised to 24.8 tons per mill hour.

Two electrically controlled air-operated slurry pumps carry the slurry from the mills to 8 slurry tanks. Most of the air used for agitation is exhaust air from the pumps. Figure 2 shows a flow sheet of the plant from the crusher to the slurry tanks.

WAGE SCALE

Following is a table showing the number and type of workmen employed and the wages paid per hour:

Position	Number of men	Wages per hour
Superintendent	1	$1.25
Well driller	1	.8325
Helper	1	.625
Shovel operators	3	1.00
Pitmen	3	.5625
Jackhammermen	3	.5625
Powderman	1	.625
Locomotive engineers.	2	.70
Brakemen	2	.625
Primary crusher	1	.70
Secondary crusher ...	1	.625

SAFETY ORGANIZATION

The safety work is supervised by a safety committee and safety engineer. Meetings are held regularly and precautions discussed, but the most effective work is done by the men themselves. They discuss safety at all times and suggestions made by them are enforced, as they are invariably excellent.

Accident record of the plant since 1928

Year	Man-hours of exposure	Frequency: Number of accidents	Frequency: Accidents per 1,000,000 man-hours	Frequency: Days lost	Severity: Days lost per 1,000 man-hours
1928	330,684	12	1.362	464	1,403
1929	357,000	7	.196	103	.289
1930	235,820	3	.127	131	.556
1931	176,043	0	0	0	0

ADMINISTRATION ORGANIZATION

Figure 3 shows the organization chart for the entire plant.

Table 1. - Summary of Costs

Calaveras Cement Co. Period covered: Jan. 1 to Dec. 31, 1930.

Total material loaded during period: Stone ...... 175,612 tons (2000 lbs.)
Stone ground 180,827 tons.

Operating costs per dry ton of stone mined and ground

	Labor 1/	Power	Explo- sives	Other supplies	Total
Mining:					
Drilling	$0.02967	$0.00222	-	$0.00071	$0.03260
Primary blasting	.00313	-	$0.04347	.00001	.04661
Secondary blasting	.03212	-	.00964	.00360	.04536
Loading	.06008	.01112	-	.00819	.07939
Transportation	.05936	-	-	.02662	.08598
Crushing 2/	.06825	.01267	-	3/.05358	.13450
Grinding 2/	.11454	.14686	-	.09983	.36123
Miscellaneous plant	.12645	-	-	-	.12645
Total direct operating costs.	0.49360	0.17287	$0.05311	0.19254	0.91212
Depreciation	.24469	-	-	-	.24469
Depletion	.02801	-	-	-	.02801
Total operating cost	-	-	-	-	1.18482

1/ Includes cost of workmen's compensation insurance.

2/ Elevating and conveying power costs are included in both crushing and grinding power costs. The estimated power cost of elevating and conveying is $0.00764 per ton, based on the same power distribution as in Table 2.

3/ Includes $0.04435, cost of new main frame of 42-inch gyratory crusher.

Table 2. - Summary of costs in units of labor, power, and supplies

Period covered: Jan. 1 to Dec. 31, 1930.

Material loaded during period: Stone, 175,612 tons.

	Mining	Crushing	Total
A. Labor (man-hours per ton):			
Well drilling and secondary drilling	0.0556	-	0.0556
Blasting	.0216	-	.0216
Loading	.0713	-	.0713
Haulage	.0773	-	.0773
Crushing	-	0.0862	.0862
Total labor	0.2258	0.0862	0.3120
Average tons produced per man per shift .	-	-	25.64
Labor, per cent of total cost	-	-	39.56
B. Power and supplies:			
Explosives (lbs. per ton)	-	-	0.3687
Total power (kw.h. per ton)	-	-	23.6040
Shovels	1.5183	-	1.5183
Drills	.3031	-	.3031
Crushing	-	1.4883	1.4883
Elevating and conveying	-	1.0432	1.0432
Compeb mills	-	19.2511	19.2511

Table 3. - Detailed average shovel costs, direct operation

Period covered: Jan. 1, to Dec. 31, 1930.

Types of shovel: (50-B; size of dipper, 2 yards.
(60-C; size of dipper, $2\frac{1}{2}$ yards.

Stone loaded: 175,612 tons.

	Total cost	Cost per ton
Superintendence	$1,075.02	$0.00612
Engineers	4,656.93	.02652
Pitmen	2,631.04	.01498
Other operating labor	779.18	.00443
Total operating labor	9,142.17	.05205
Fuel or power	1,952.85	.01112
Operating supplies, grease, and lubricants	781.99	.00445
Total supplies	2,734.84	.01557
Shovel repair labor	1,409.82	.00803
Repair supplies	656.38	.00374
Total repairs	2,066.20	.01177
Total shovel operation	13,943.21	0.07939

Table 4. - Detailed summary of costs - direct operation

Period covered: Jan. 1 to Dec. 31, 1930.

Types of shovel: (50-B; size of dipper, 2 yards.
(60-C; size of dipper, $2\frac{1}{2}$ yards.

Stone loaded: 175,612 tons.

	Total cost	Cost per ton
Total, all shovels	$13,943.21	$0.07939
Drilling:		
Operating labor	4,805.00	.02736
Air, gas, power	390.56	.00222
Operating supplies	34.13	.00019
Repair labor	404.59	.00231
Repair supplies	90.78	.00052
Total drilling	5,725.06	0.03260
Blasting:		
Labor	6,190.63	.03525
Explosives	9,325.77	.05311
Other supplies	634.04	.00361
Total blasting	16,150.44	0.09197
Haulage:		
Locomotives and cars, operating	7,764.32	.04421
Locomotives and cars, repairs	2,658.90	.01514
Truck maintenance	4,677.06	.02663
Total haulage	15,100.28	0.08598
Grand total	50,918.99	0.28994

[illegible]

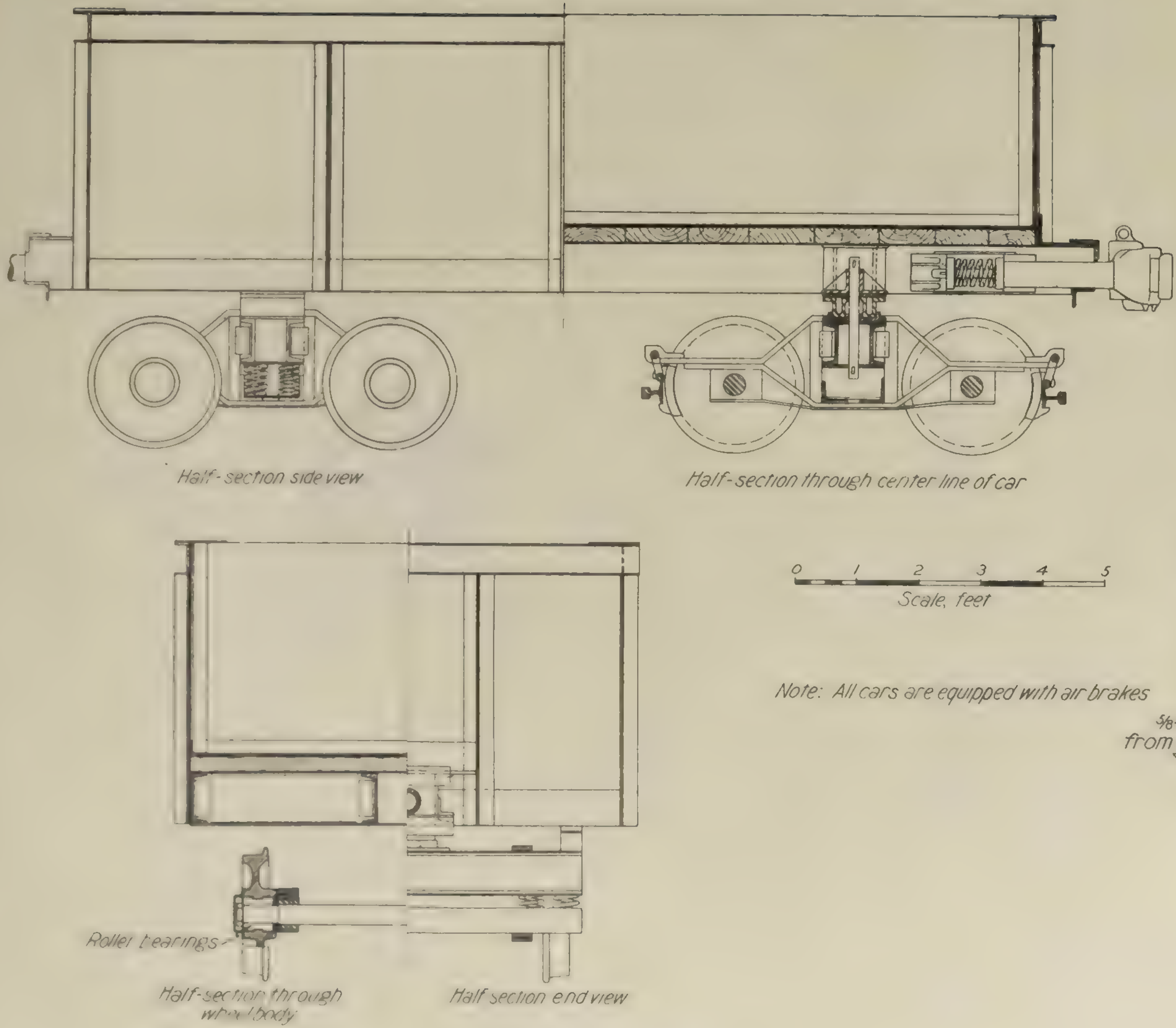

Figure 4.-Rotary-dump quarry car

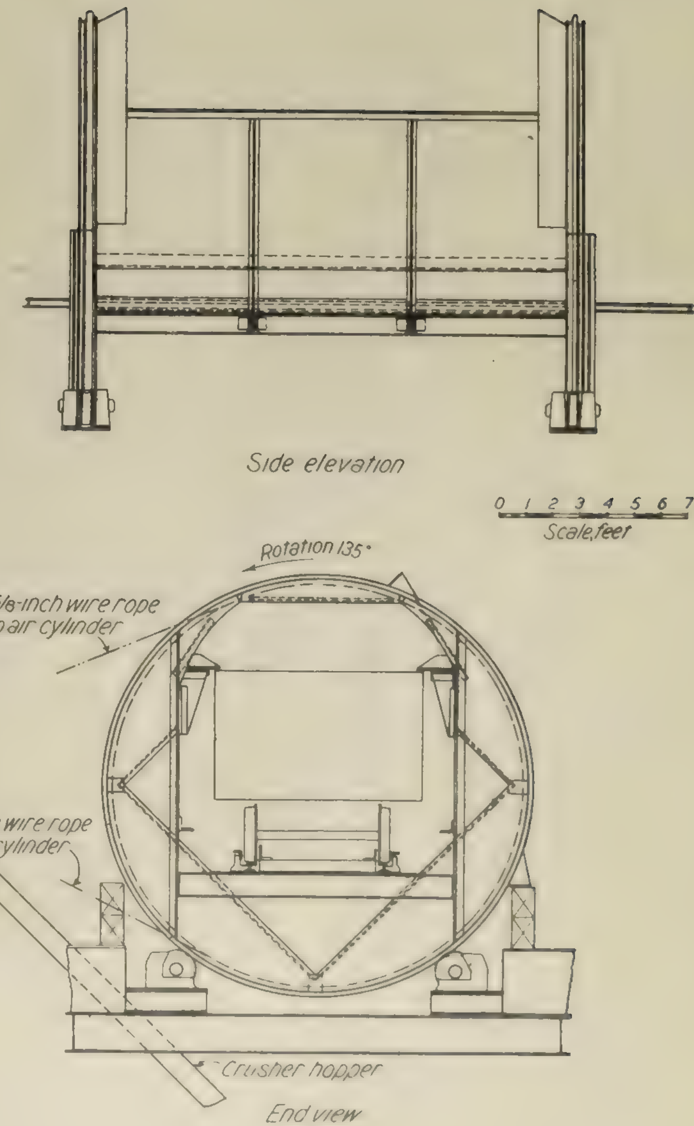

Figure 5.-Rotary car-dumper

I. C. 6611

April, 1932

(Revised June, 1932.)

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SMALL-SCALE PLACER-MINING METHODS

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SMALL-SCALE PLACER-MINING METHODS[1]

By Chas. F. Jackson[2] and John B. Knaebel[3]

INTRODUCTION

During the current business depression the Bureau of Mines has received many inquiries concerning placer mining, the prospects for men of small means to make wages by small-scale placer operations, the locations of placer deposits in the United States, methods of working placers on a small scale, and allied subjects. It has therefore seemed desirable to prepare a circular covering in nontechnical language, the principal questions which have been referred to the Bureau of Mines for answer. There are a number of bureau publications dealing with placer mining, most of which are now out of print, but which may be consulted in the larger libraries. Other articles on placer mining have appeared from time to time in the technical press and several good textbooks have been written on the subject. A partial bibliography of this literature appears at the end of this paper. A recent pamphlet published by the Idaho Bureau of Mines and Geology[4] to assist the prospector and inexperienced small operator has received wide and favorable attention and is recommended as a useful reference.

In the present paper no attempt will be made to cover the subject of placer mining in a comprehensive manner, but discussion will be confined to elementary questions of particular interest to persons of limited experience.

OUTLOOK FOR SUCCESS OF SMALL PLACER OPERATIONS IN THE UNITED STATES

During recent months articles have appeared in the press encouraging the belief that wages can be easily earned in the old placer fields of this country by rewashing the sands and gravels of the creek beds and of their bars and benches. The amount of gold recovered by small operators and sold to the Government assay offices in some of the western districts in 1931 was about double that of the years immediately preceding. To encourage the small operator the assay offices were recently authorized to purchase gold in minimum quantities of 40 dollars worth, whereas the previous minimum was 100 dollars. This regulation requires, however, that the seller state the source of his gold.

It may be said at the outset that the old-timers knew their business well and were thorough. Many attempts to rework the early placer mines have demonstrated that as a rule they left little behind which could be profitably worked by small-scale methods. It is quite possible in some instances, however, that during the years since the placers were worked, shifting stream currents and wave action on beach placers may have worked over the material and produced new local concentrations of gold. Such concentrations are apt to be small and scattered, however, and the search for them long and arduous.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used. "Reprinted from U. S. Bureau of Mines Information Circular 6611."

2 - Principal mining engineer, U. S. Bureau of Mines.

3 - Assistant mining engineer, U. S. Bureau of Mines.

4 - Staley, W. W., Elementary Methods of Placer Mining: Pamphlet No. 35, Idaho Bureau of Mines and Geology, University of Idaho, Moscow, Idaho, May, 1931, 20 pp.

A few experienced placer miners, aided by an element of good luck, are probably doing very well at small-scale placer mining. The vast majority, however, are probably not averaging $2 per day.

Walter W. Bradley, state mineralogist of California, recently estimated that 10,000 or more itinerant men and women prospected in the gold region of northern California during 1931 and took about $1,000,000 in gold. He estimated that they worked an average of 90 days during the year and averaged a little more than $1 per day.

In Colorado the daily earnings are estimated by another authority to be somewhat higher.

There have been no important new discoveries of gold placers in this country in recent years and it is unlikely that there will be. In a country untouched by railroads and highways, the streams frequently afford the easiest avenues of travel, and it is unlikely that important placer deposits have been overlooked by prospectors (who always follow the streams) and by other travelers in these regions. That small and comparatively low-grade deposits still remain undiscovered is quite possible and the search for them is a laudable effort. The experienced prospector has the best chances for success, while there is little chance for a man unfamiliar with placer mining, unused to hardship and lacking experience in "roughing it."

In spite of the low grade of material left behind by the early miners, the best opportunity for the novice to pan wages would still seem to be in the districts in or adjoining the old workings. The accompanying maps, Figures 2 to 16, inclusive, show the locations (indicated by black circles) in the United States from which placer gold is known to have been won in the past (figure 1 omitted). The fact that placer gold has been recovered from these districts does not mean that profitable operations can now be conducted therein.

GEOLOGY AND TYPES OF PLACER DEPOSITS

A lengthy discussion of the origin of placers and their occurrence is beyond the scope of the present paper. Some knowledge of these subjects is essential, however, to intelligent search for and operation of placers. Although other valuable substances than gold, such as platinum, tin, tungsten, gems, and minerals of the rare metals are found in placers, the present paper deals exclusively with gold placers.

Types of Placer Deposits

There are two general types of placer deposits: residual placers and transported or alluvial placers. In both types the original source of the gold was in lodes or veins in solid rock. In both types the gold has been freed from the inclosing rock through weathering agencies which have resulted in the decomposition of the rock and the partial removal of some of its constituents by solution or mechanical means. Since gold is unaffected by the usual weathering agencies it remains in its original form.

In the case of residual placers the gold and most of the associated, decomposed rock is left in its approximate original position or very close thereto. Placers of this type have not been as important sources of gold production as have the transported placers. In a recent visit to western gold districts the author noted old workings in this type of deposit at Central City, Colo., where a considerable area of altered gold-bearing gneiss had been worked on the hillside and in a narrow gulch close to the lode which is now being mined by open-cut and underground methods. Much of the gold produced in the Appalachian region was recovered from residual placers.

Transported placers result from the removal of gold-bearing rock débris from its original source in the lode by natural agencies, principally running water, and the redeposition

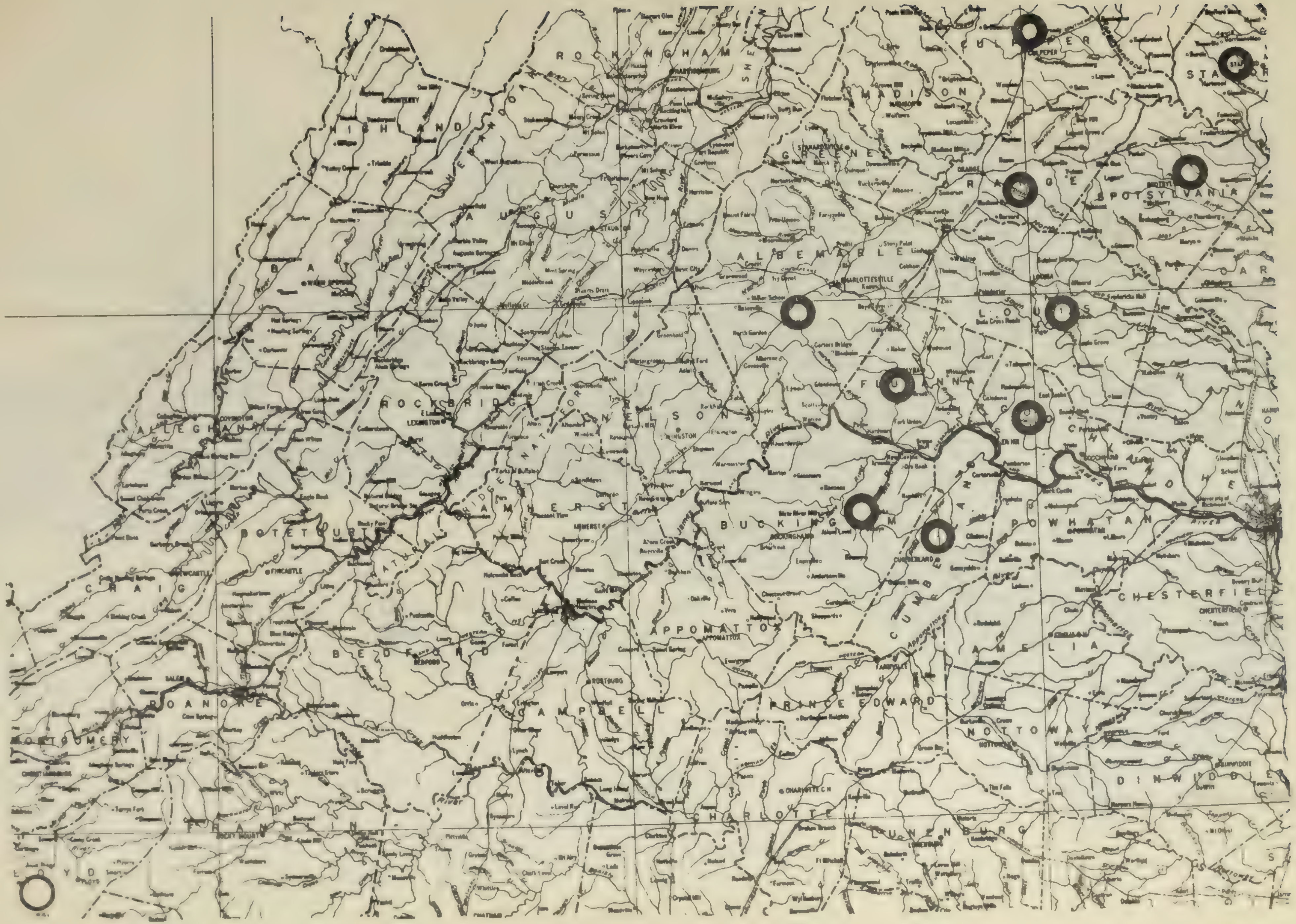


Figure 2.- Placer mining districts (circled) in Virginia

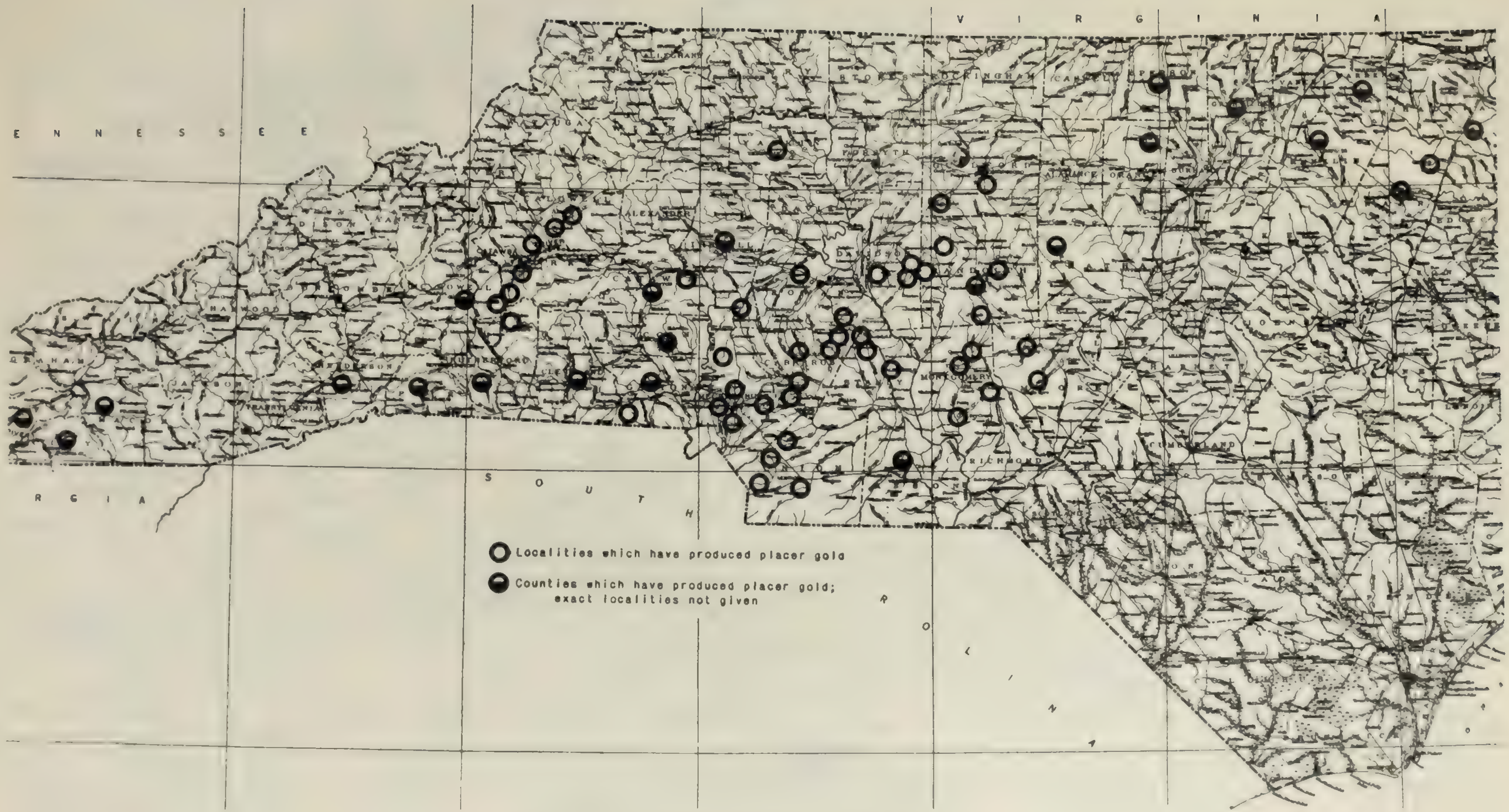


Figure 3.- Placer mining districts in North Carolina

Figure 4.- Placer mining districts (circled) in South Carolina

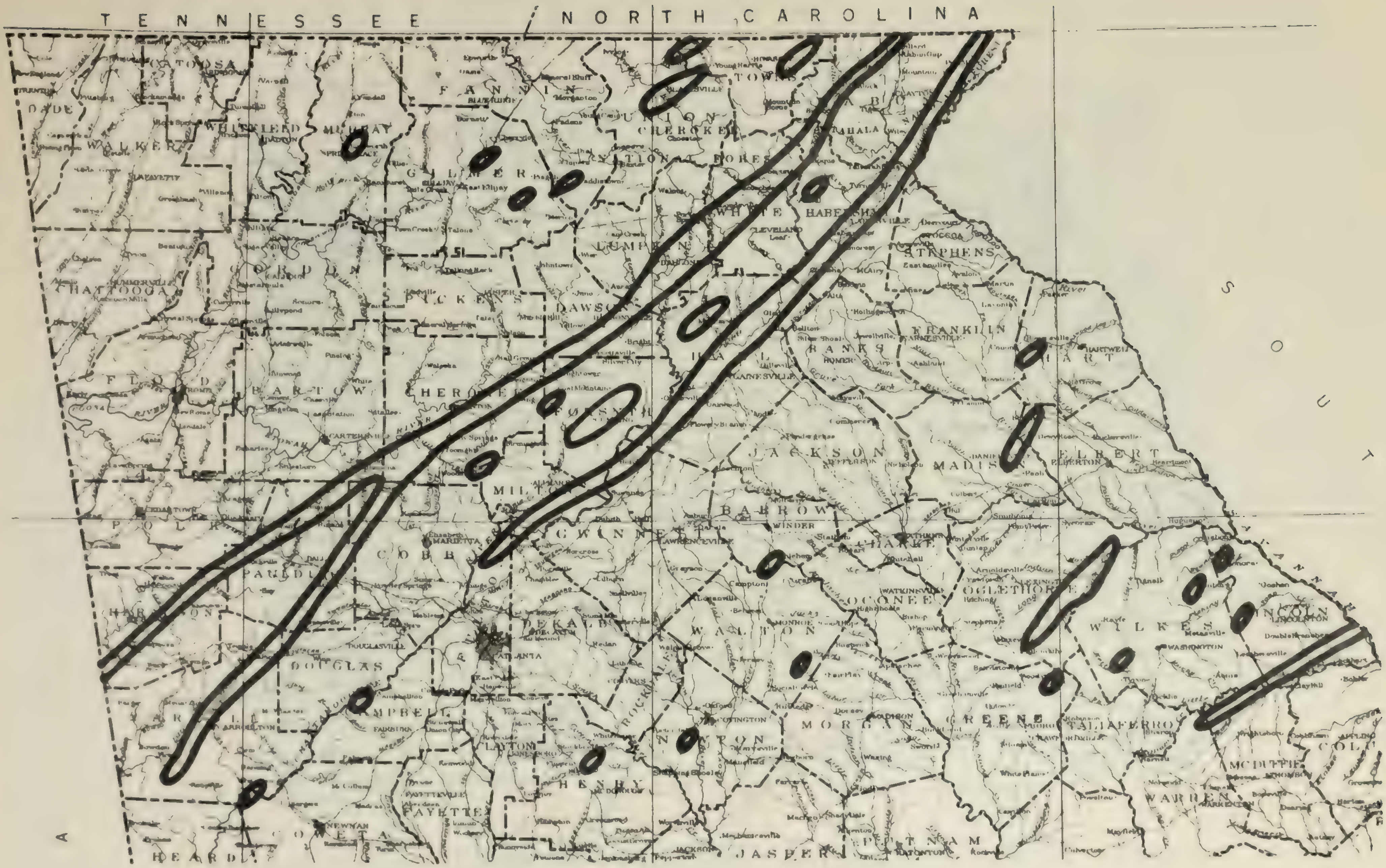


Figure 5.- Placer mining districts (darkened) in Georgia

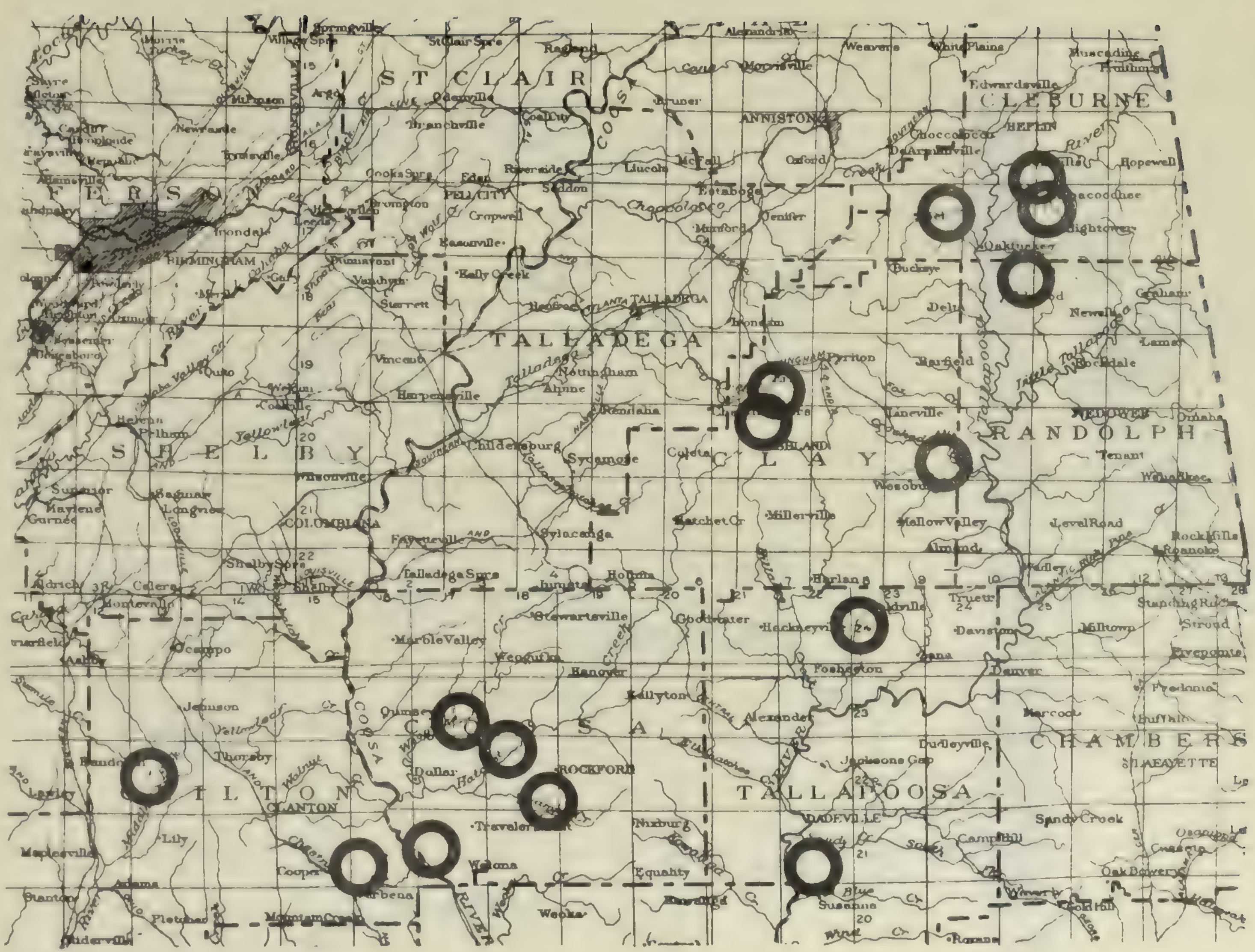


Figure 6.- Placer mining districts (circled) in Alabama

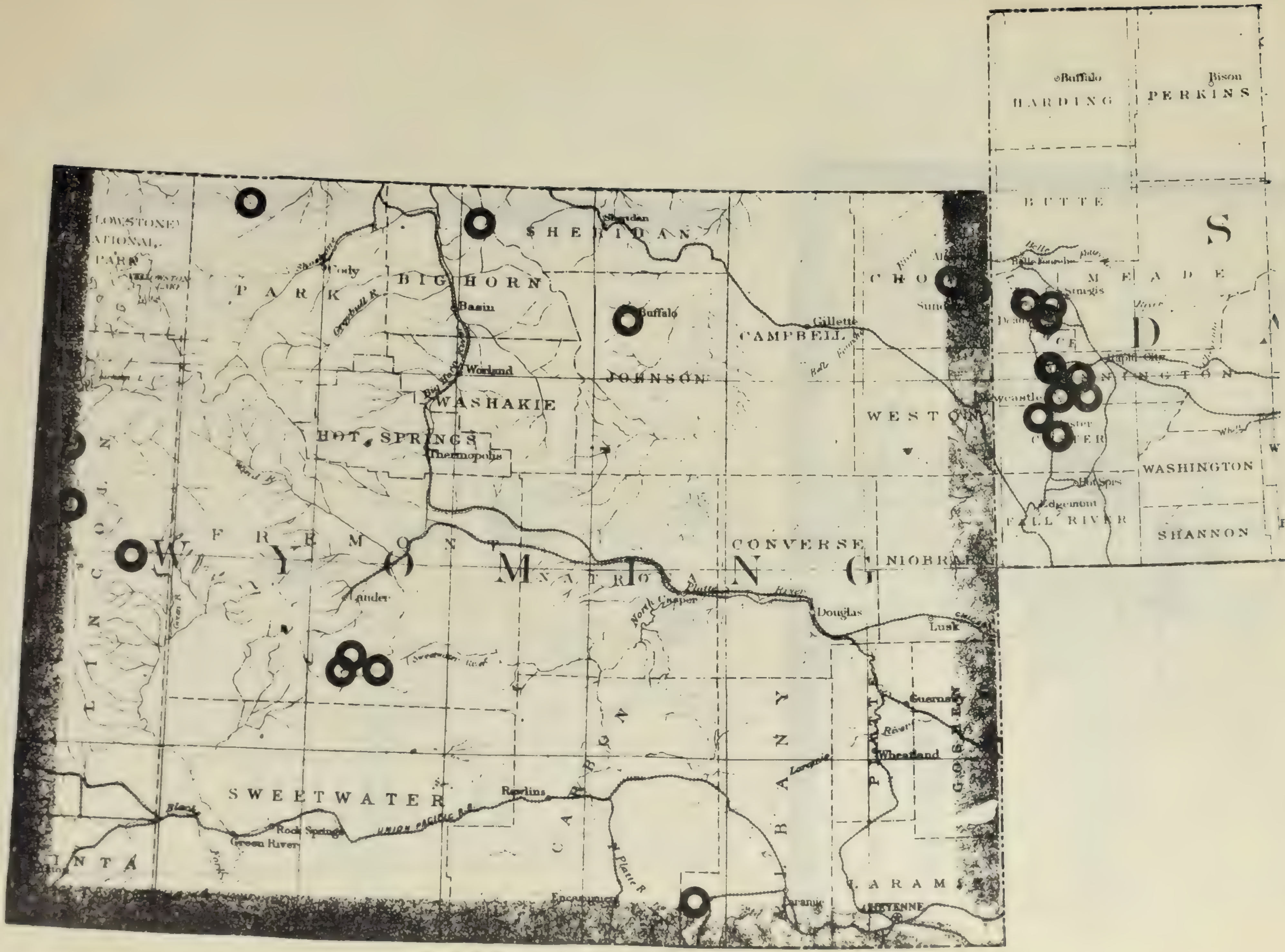


Figure 7.- Placer mining districts (circled) in Wyoming and western South Dakota

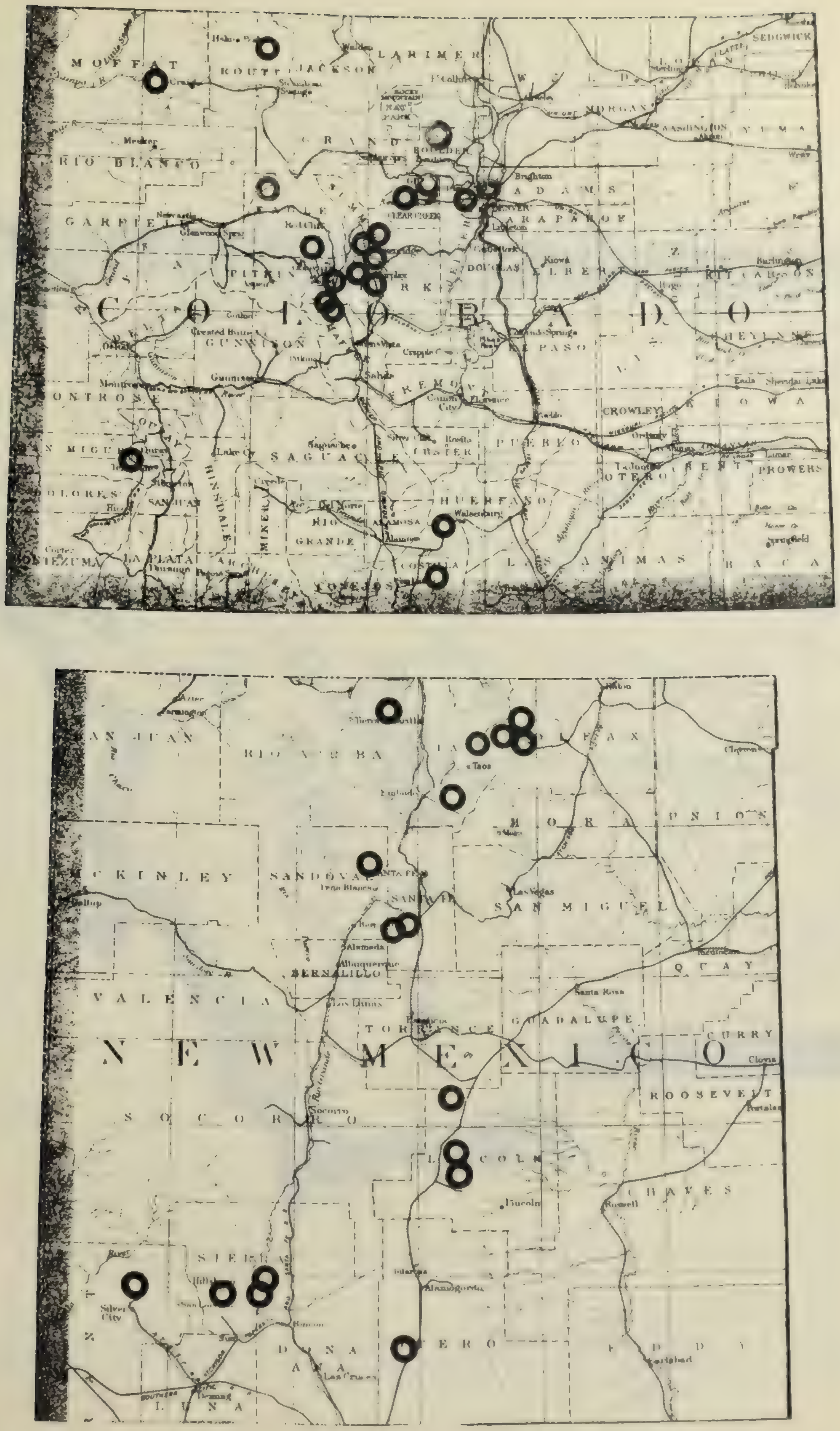


Figure 8.- Placer mining districts (circled) in Colorado and New Mexico

Figure 9.- Placer mining districts (circled) in Montana

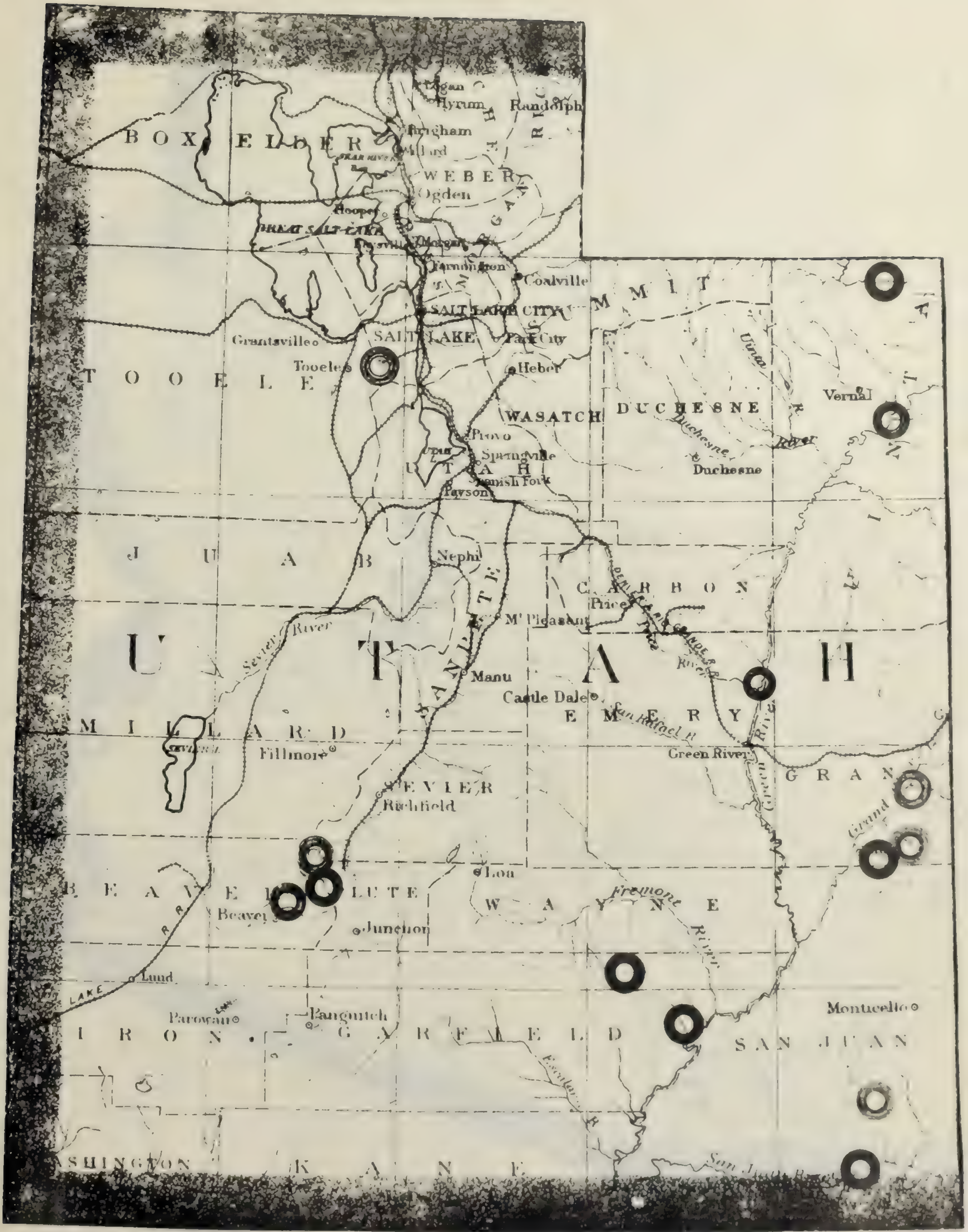
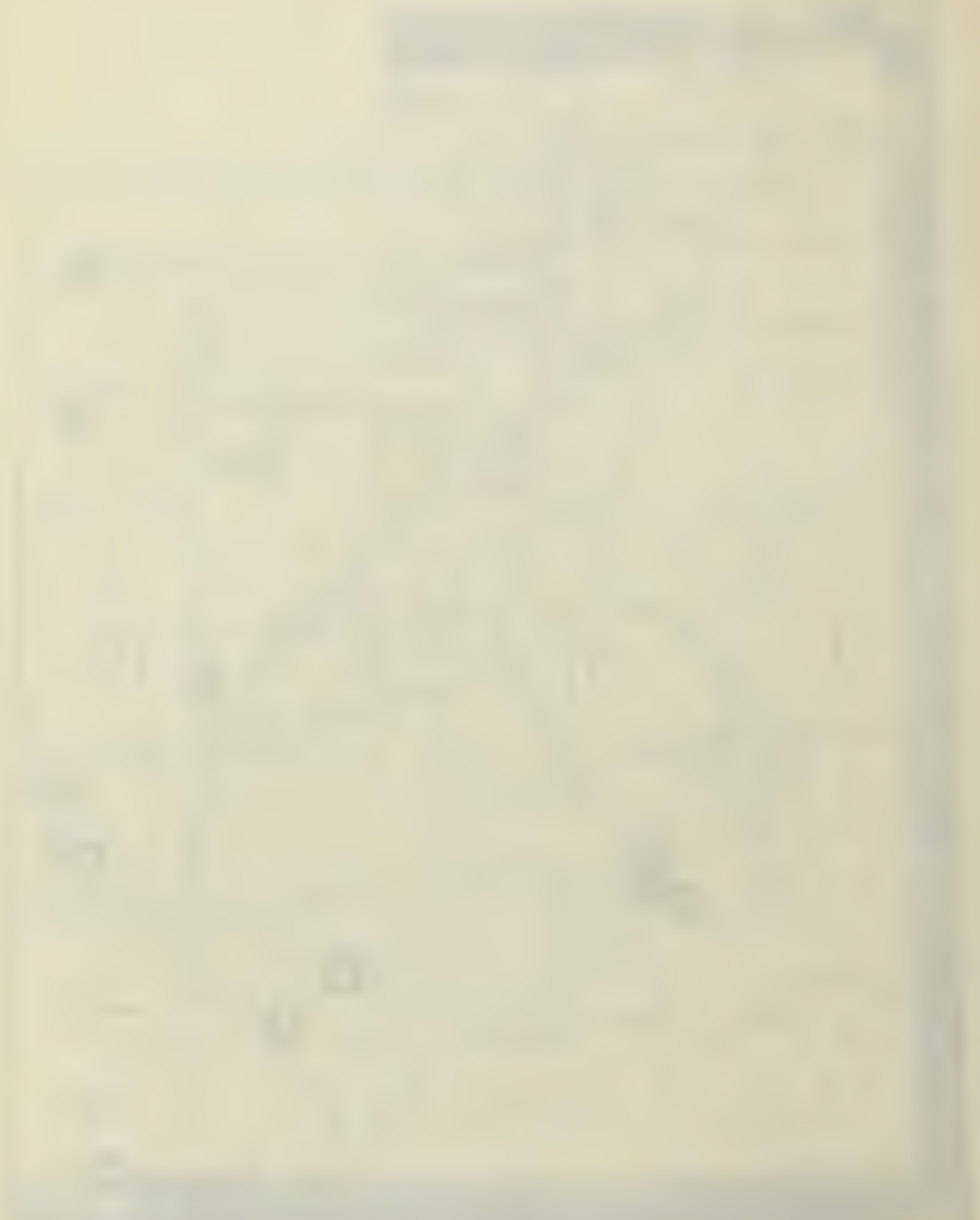


Figure 10.- Placer mining districts (circled) in Utah



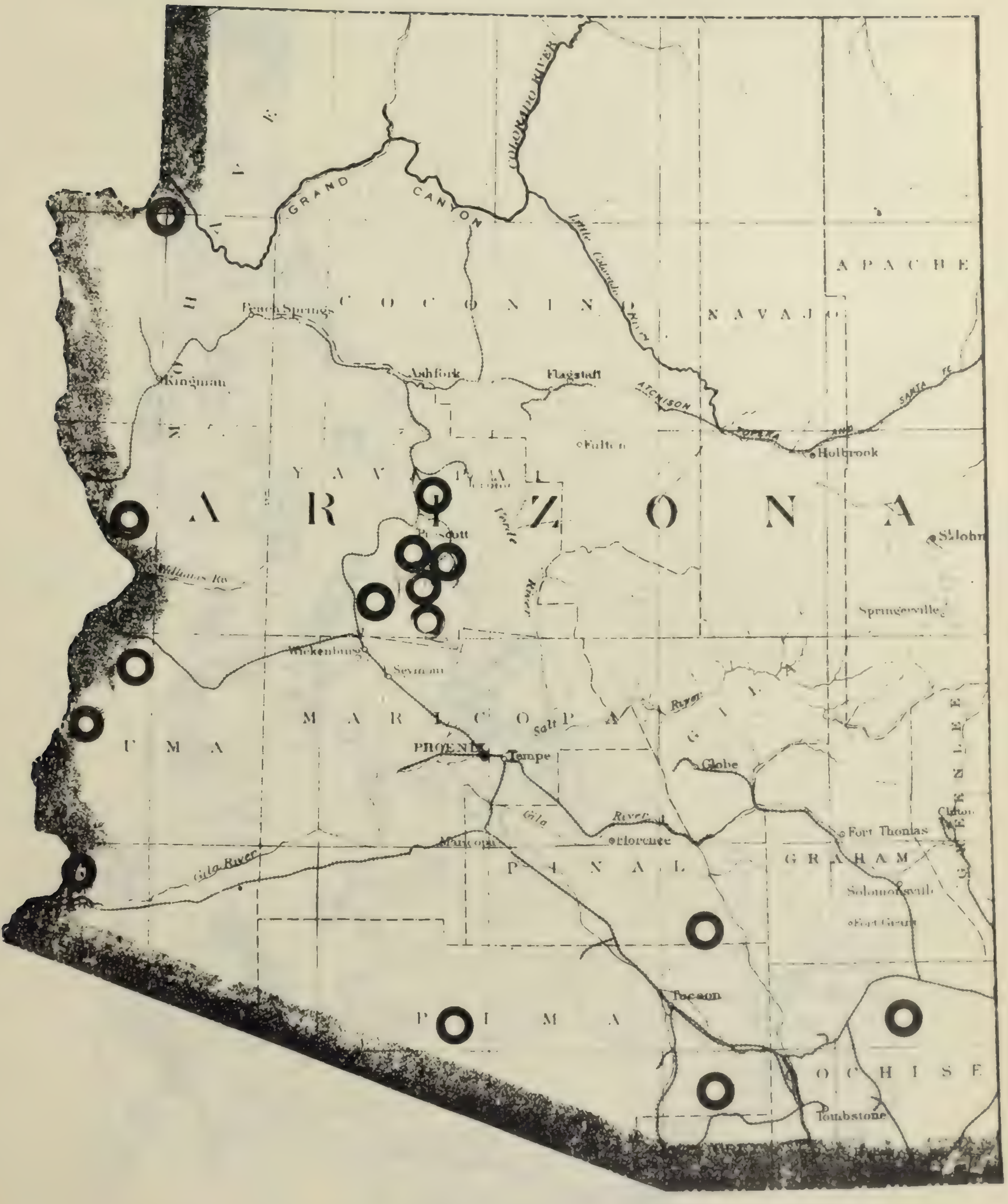


Figure 11.- Placer mining districts (circled) in Arizona

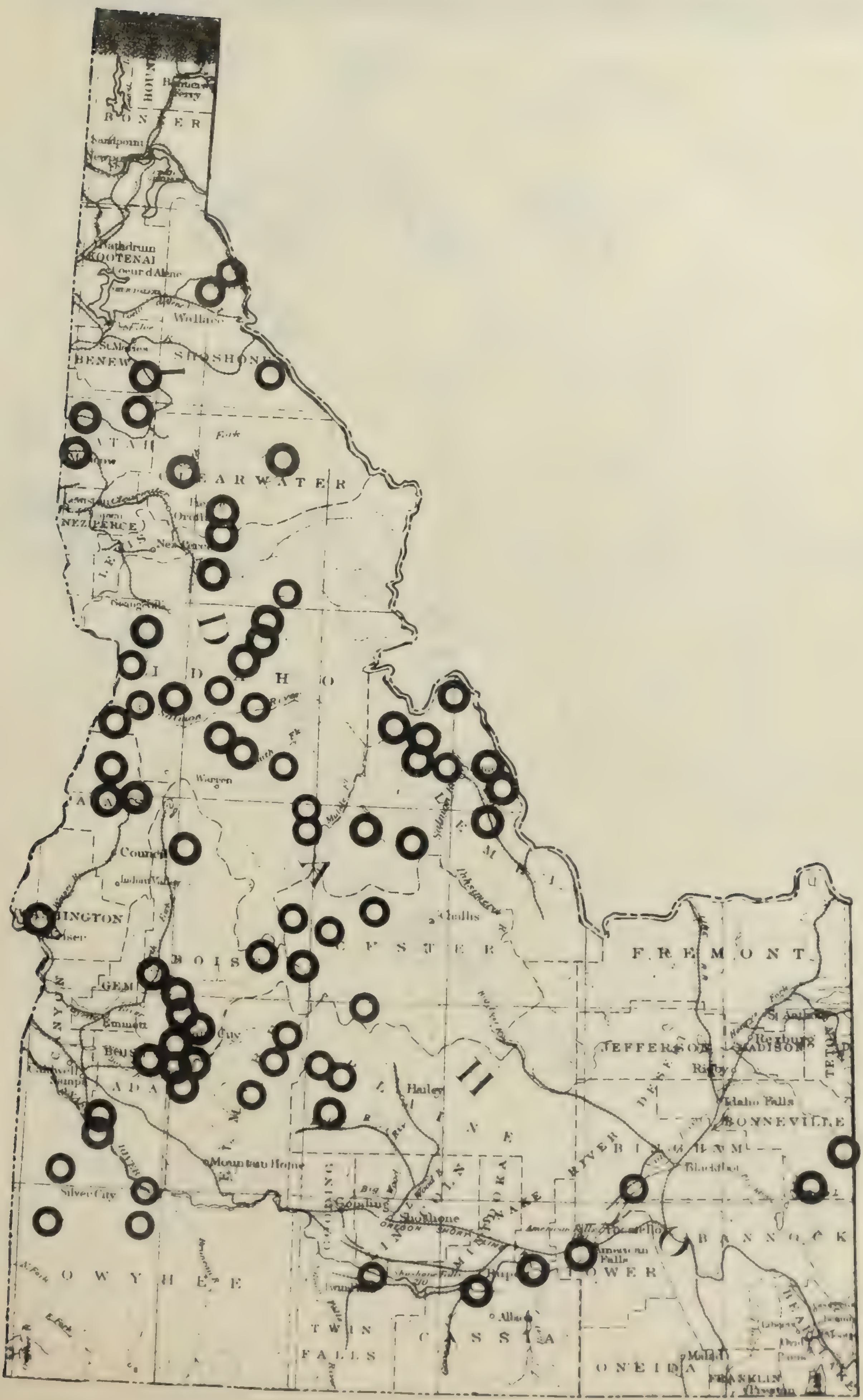


Figure 12.- Placer mining districts (circled) in Idaho

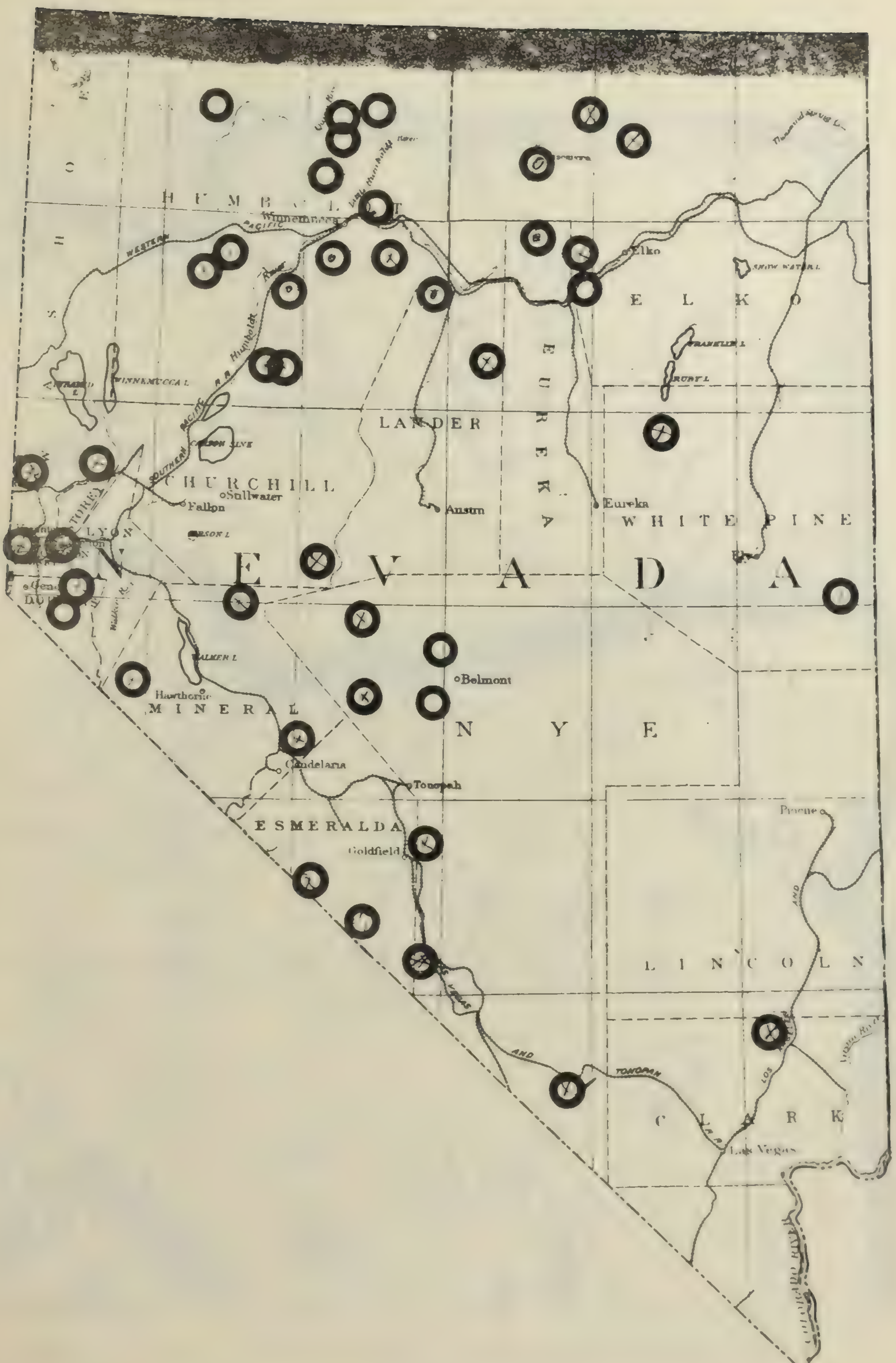


Figure 13 - Placer mining districts (circled) in Nevada

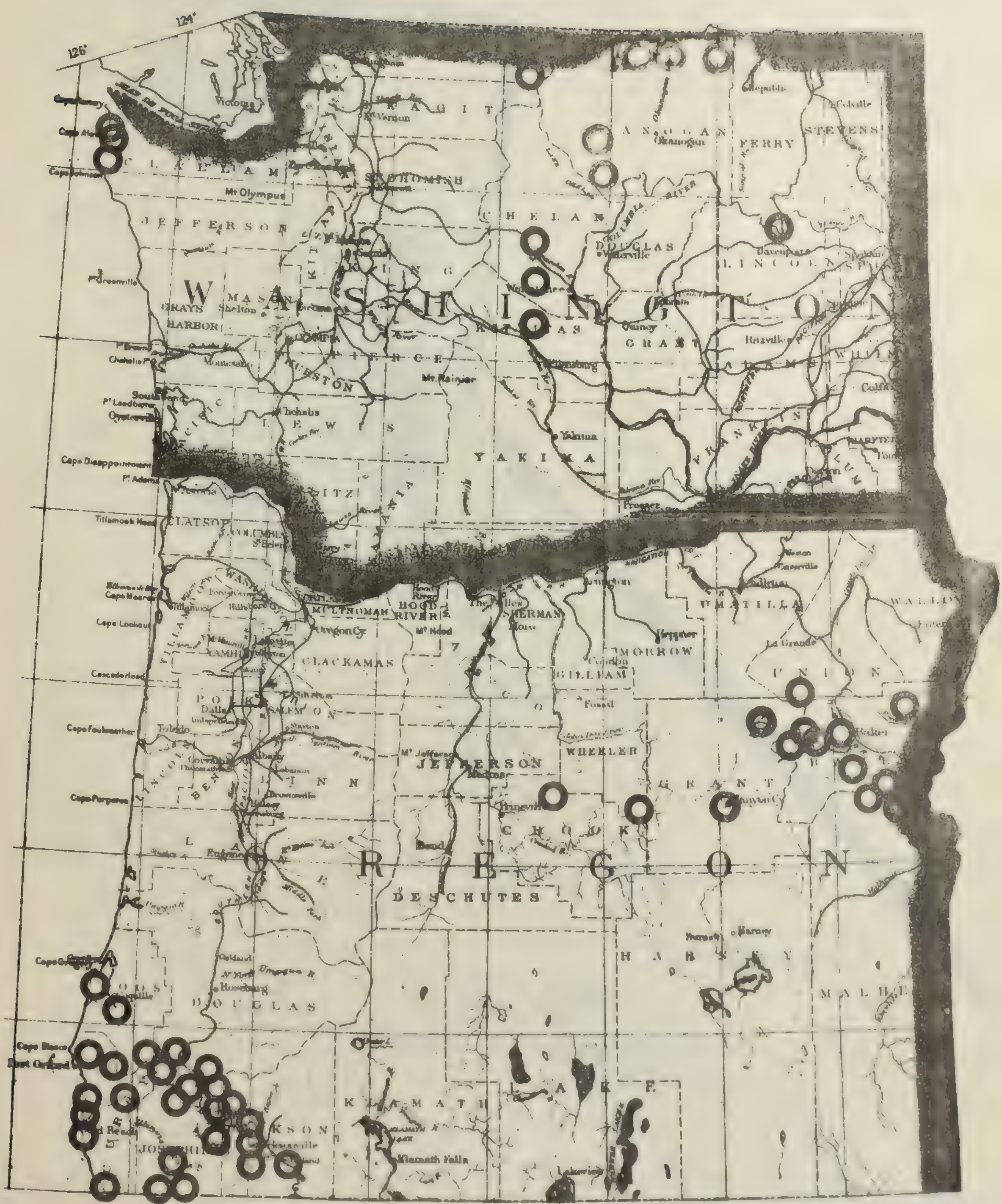


Figure 14.- Placer mining districts (circled) in Washington and Oregon

Figure 15.- Placer mining districts (circled) in California

DISTRICTS

No.		No.		No.		No.	
1	Nome	15.	Chandalar	29.	Gold Hill-Fort Gibbon	43.	Delta
2	Solomon.	16	Wade Hampton Marshall.	30.	Hot Springs.	44	Chistochina Chisna
3.	Bluff	17.	Goodnews Bay	31	Rampart	45.	Nabesna
4.	Council.	18.	Kuskokwim Tuluksak-Aniak	32.	Tolovana-Livengood	46	Chisana Shushana
5	Koyuk Dime Cr	19	Kuskokwim-Georgetown	33.	Fairbanks	47	Nizina
6	Fairhaven Candle.	20	Kuskokwim-Mt McKinley.	34.	Chena-Salchaket	48.	Nelchina
7	Fairhaven-Inmachuck.	21.	Iditarod.	35.	Bonnifield.	49.	Girdwood Crow Cr.
8	Kougarok	22.	Innoko	36.	Richardson-Tenderfoot.	50	Hope Sunrise
9	Pt. Clarence-Teller.	23.	Tolstol.	37.	Circle.	51	Kodiak beaches
10	Noatak.	24.	Lake Clarke	38.	Woodchopper-Coal Cr.	52	Yakataga beaches
11	Kobuk-Squirrel River.	25.	Swentna-Rainy Pass.	39.	Seventymile.	53	Yakutat beaches
12	Kobuk Shungnak	26.	Yentna-Cache Cr	40.	Eagle	54.	Lituya beaches
13	Koyukuk Hughes.	27.	Kantishna.	41.	Fortymile.	55.	Juneau
14.	Koyukuk Nolan	28.	Ruby.	42.	Valdez Cr.	56.	Porcupine

Figure 16.- Placer mining districts (circled) in Alaska, after Wimmler

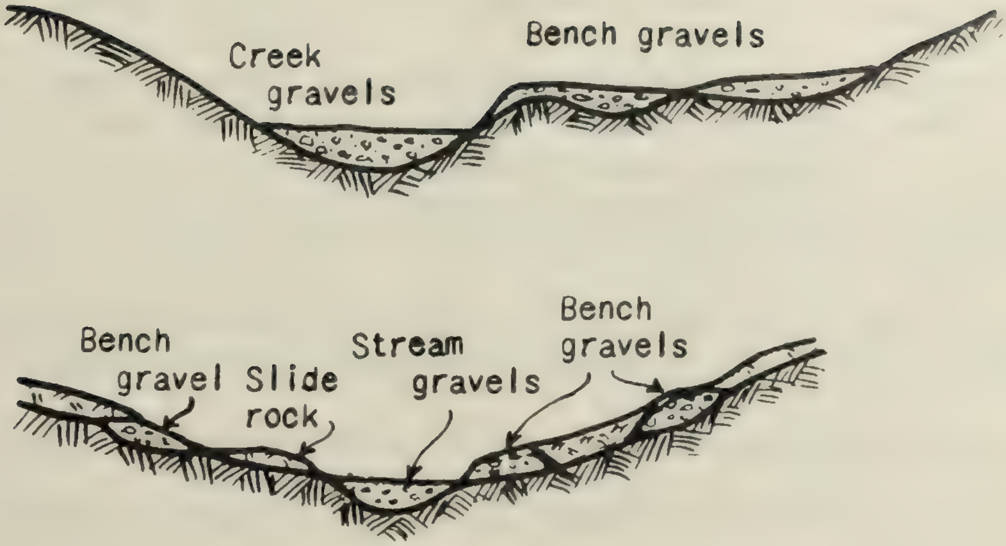


Figure 17.- Creek and bench gravels (from U. S. Geological Survey)

of the material at some distance therefrom. During transportation the rock is broken up and worn away, further liberating the gold. Another important action of moving water without which workable concentrations of placer gold could not occur, is its sorting action, which separates the heavier gold from the light material. Briefly then, weathering and other erosional forces break down the lode material and inclosing rocks, the débris moves down the hillsides toward the stream beds, and the streams carry it along. The lighter and finer material is quickly washed out and carried away, while the heavier material, including the gold, and the larger material is deposited in the stream channels at places where the velocity of the water is sufficiently reduced. The largest and heaviest material will obviously be deposited first (that is, nearest its source in the lode) and the lighter and finer material will be transported farther. Re-sorting of placers may take place because of changes in stream courses or velocities of currents, and sometimes the placers may be worked over several times by natural agencies before the gold reaches a final resting place.

Classification of Transported Placer Deposits

The following classification by Brooks[5] is based largely upon the present position of the deposits:

Creek placers.- Gravel deposits in the beds and intermediate flood plains of small streams (fig. 17).

Bench placers.- Gravel deposits in ancient stream channels and flood plains which stand from 50 to several hundred feet above the present streams (fig. 17). They represent the remnants of the stream beds during earlier stages of stream development.

Hillside placers.- A group of gravel deposits intermediate between the creek and bench placers. Their bedrock is slightly above the creek bed and the surface topography shows no indication of benching.

River-bar placers.- Placers on gravel flats in or adjacent to the beds of large streams.

Gravel-plain placers.- Placers found in the gravels of the coastal or other lowland plains.

Sea-beach placers.- Placers reconcentrated from the coastal-plain gravels by the waves along the seashore.

Ancient beach placers.- Deposits found on the coastal plain along a line of elevated benches.

Lake-bed placers.- Placers accumulated in the beds of present or ancient lakes that were generally formed by landslides or glacial damming.

The creek placers have been the most productive of high-grade gravel. The gold is usually concentrated to a large extent on and just above bedrock. Enriched pockets are apt to be found in depressions in the bedrock and if the rock is decomposed or fractured, gold will have settled into the cracks and crevices. It is therefore essential in working to excavate to bedrock in every case, and often it is necessary to take up some of the rock to recover the gold accumulated in the crevices.

The presence of beds of clay or "hardpan" in placer deposits may have a marked influence on the distribution of the gold. They form impervious layers on which concentration of gold takes place and act to prevent the gold from working below them. This does not mean that no gold will be found below such beds, for gold may have been deposited at lower horizons before the clay beds were formed.

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5 - Brooks, A. H., The Mineral Deposits of Alaska: Mineral Resources of Alaska, 1913, U. S. Geol. Survey Bull. 592, 1914, pp. 25-32.

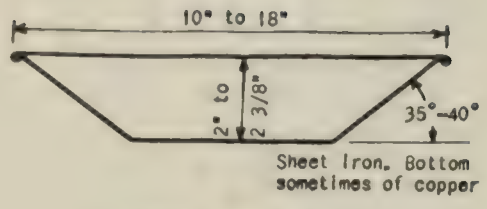

Figure 18.- Gold pan

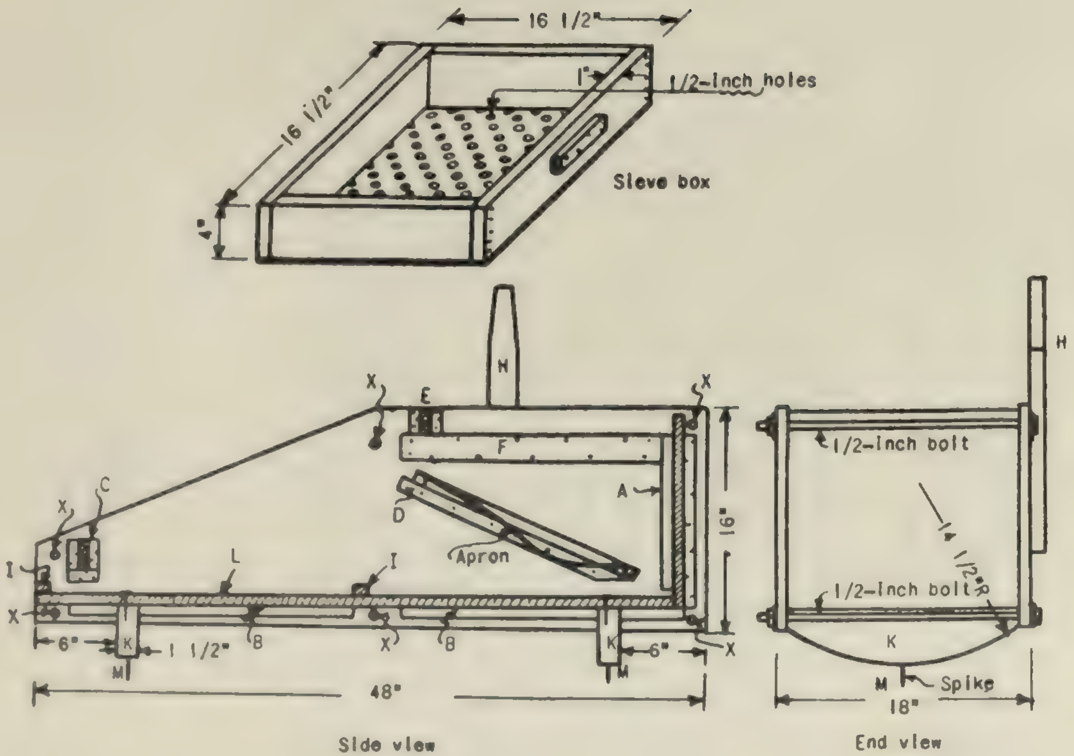

Figure 19.- Sketch of knockdown rocker and view of sieve box (from Engineering and Mining Journal)

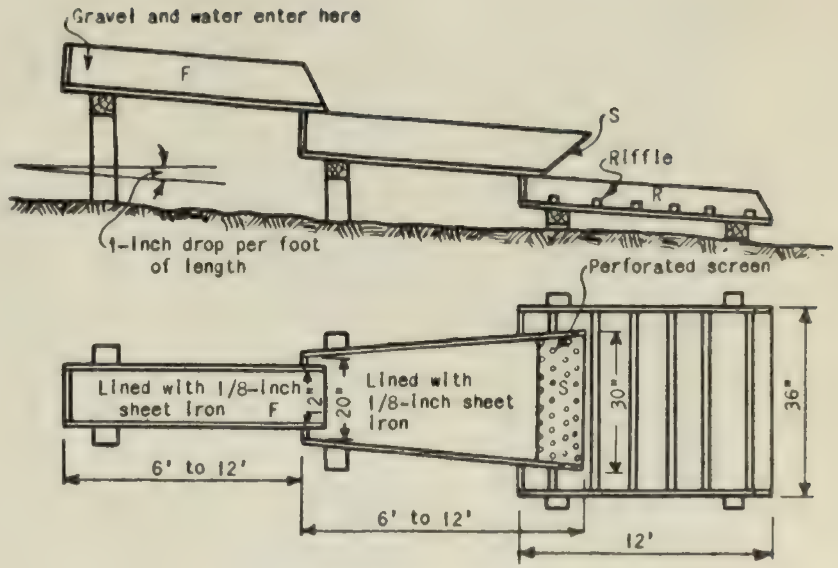

Figure 20.- Long tom

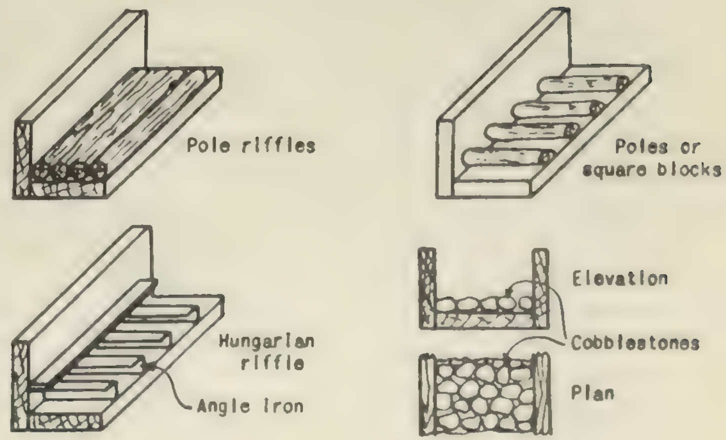

Figure 21.- Common types of riffles

Nuggets.-

Coarse gold - that which remains on a 10-mesh screen (10 openings per linear inch).

Medium gold - that which remains on a 20-mesh screen but passes a 10-mesh screen (average 2,200 colors per ounce).

Fine gold - that which passes a 20-mesh screen and remains on a 40-mesh screen (average 12,000 colors per ounce).

Very fine gold - that which passes a 40-mesh screen (average 40,000 colors per ounce).

Flour gold.-

170 colors to 1 cent (314,500 per ounce).

280 colors to 1 cent (436,900 per ounce).

500 colors to 1 cent (885,000 per ounce).

Pure gold is worth $20.67 per ounce. Gold as it occurs in nature is usually alloyed with varying proportions of silver, and the average value of placer gold may be roughly taken at about $18.50 per ounce. Medium gold as above defined is therefore worth about $\frac{84}{100}$ of a cent per color, and fine gold about $\frac{15}{100}$ of a cent per color.

Gold in the coarser sizes is much easier to recover. Fine or flour gold is apt to float off with the light material, either free or tied up in fragments and fine pieces thereof.

PROSPECTING

The first search for placer gold is usually confined to stream beds, their bars, and to tributary gulches, since, as previously pointed out, the streams and their tributaries are the principal agencies in the formation of placers. Even though the most valuable deposits may be in benches high up on the slopes, their presence would almost invariably be evidenced by showings of gold along the streams below.

In prospecting for gold, either in a small or a large way, the gold pan is an indispensable tool. Figure 18 shows the ordinary gold pan, which is usually made of sheet iron. Ordinary iron is the best material to use, since the colors seem to adhere to it better than to surfaces of other materials, and since it provides a surface upon which the colors are easily visible. Some gold pans are provided with copper bottoms. Both copper and gold amalgamate readily with mercury and if mercury is rubbed on the copper bottom, much of the fine gold which otherwise might be washed away will be retained by amalgamation with the mercury. The inner surface of the pan should be kept clean, bright, and free from grease.

In prospecting along a stream the prospector will pan the gravel at various points, selecting particularly such places as show evidences of concentration of heavy minerals by the presence of black sands. Since the gold and heavy sands will ordinarily be found concentrated on bedrock, the prospector will investigate exposures of bed-rock in and along the stream, especially depressions therein where these minerals may be caught. In addition, excavations to bedrock will be made in the gravels of the stream bed and its bars and along its banks.

Panning

The material to be tested is dumped into the pan, filling it about two-thirds full. The operation of panning is simple but requires a knack only developed by practice and is difficult to describe clearly by words alone. The pan of material is held under water and is

mixed and kneaded with the hands, breaking up lumps of clay and other easily broken material. If the pan is held in running water, some of the lightest and finest material will immediately wash away. The rocks and pebbles are picked out and discarded, and after all the lumps of soft material are broken up to free any gold trapped in them, the regular panning operation is begun. The pan is raised to just below the surface of the water and shaken vigorously from side to side with a slightly circular motion which keeps the lighter material in suspension and works it out of the pan. The pan is held tilted slightly away from the operator. The motion of the pan serves to concentrate the gold and heavy minerals around the edge of the bottom of the pan. The washing away of the light material is facilitated by alternately raising and lowering the far edge or lip of the pan above and below the surface of the water. The pan may be occasionally lifted entirely from the water and shaken vigorously with the usual circular motion to concentrate the gold and heavy sands and to bring the pebbles and fine light material to the top. This material may then be scraped off the lip of the pan with the thumb, thus hastening the operation of panning. The panning is continued till only the gold and heaviest minerals remain in the pan. Toward the end of the operation it may be well to finish panning in a tub of water instead of in the stream, since any gold that may be inadvertently carried away may later be recovered by repanning the contents of the tub. The final product is dried and black sand is removed with a magnet. Coarse gold can be picked out color by color and fine gold may be recovered by amalgamation with mercury. An experienced man can carefully pan about 100 pans in 10 hours, the exact amount depending upon the skill of the panner, the extent to which the gravel is cemented and whether it is clayey or not, and the size of the gold. The ordinary gold pan is estimated to hold 267 cubic inches on the average - or, putting it the other way around, 176 pans are equivalent to 1 cubic yard of material in place.[7] At Fairbanks, Alaska, most miners compute 189 pans per cubic yard. A good panner will therefore pan about 0.6 cubic yard per day, and in order to make $5 per day, the dirt would have to average about $10 per cubic yard. This would be considered rich gravel to-day. In exceptional cases a man will sometimes pan 1 cubic yard per day.

For large deposits of low-grade placer material which can only be worked profitably by mechanical means involving large capital expenditures, a thorough preliminary prospecting of the deposit is essential. This may be done by means of test pits or drill holes, depending upon the characteristics of the deposit in each case. It is not within the scope of this paper to discuss such operations, but it may be said that the gold pan is an important implement in this connection also.

In small-scale handwork, prospecting and actual mining, whether by panning, rocking, or sluicing, are conducted together as one and the same operation. That is to say, the deposit is prospected as it is mined, the work being shifted from place to place according to disclosures made during progress of the work; the results of panning are ordinarily used as a guide.

PLACER-MINING METHODS

Placer mining is conducted by both open-cut and underground methods. Underground placer mining, commonly known as "drift mining," is employed in mining buried placers. Open-cut methods may be classified on the basis of equipment employed, as follows:

7 - Peele, Robert, Mining Engineers Handbook: 1st ed., John Wiley and Sons (Inc.), New York, Vol. 1, 1918, p. 755.

1. Hand methods:
    a. Panning.
    b. Rocking.
    c. Long-toms and surf-washers.
    d. Sluicing, including ground sluicing and "booming."
2. Power methods with sluicing:
    a. Drag scraping.
    b. Hydraulic mining.
    c. Power shoveling.
    d. Dredging.
3. Dry-placer mining.

The power methods all employ sluicing in some form for the recovery of the gold, the chief distinction between power and hand methods being in the method and equipment used for excavating and conveying the material to the sluices. The sluices are the same in principle in either case. The present paper is concerned principally with hand methods.

Hand methods are applicable to small-scale operations and, since little capital expenditure is required, are suited to the small individual operator or group of operators possessed of only small means. They are in general applicable to deposits of shallow depth having only a shallow covering of barren material.

Panning

The limitations of panning as regards amount of material which can be handled and as to the grade of material which can be treated profitably have already been pointed out. Panning is slow, laborious work, but as the only tools required are pick, shovel, and pan, it is often the poor man's favorite method.

Rocking

Rocking requires little more in the way of equipment than panning and may be employed by the small operator to increase substantially the amount of gravel which can be handled in a shift. The rocker may be operated by one man, but preferably by two men, one of whom may be excavating and carrying the gravel to the rocker while the other operates the rocker, turn and turn about. The rocker may be used with the pan in prospecting work preliminary to large-scale operation, or by the small operator for actual mining operations. Purington [8] gives the duty of two men rocking steadily as 3 to 5 cubic yards per 10-hour day, the amount depending on the nature of the gravel, distance it has to be carried, etc.

Figure 19 is a sketch of a knockdown rocker [9] which may be easily constructed and requires only a small expenditure for material. Explanation of the lettered parts in the sketch follows:

A. Cleats for holding the back of the rocker.
B. Cleat for holding bottom of rocker L.
C. Cleats for holding back of rocker.
D. Cleat for holding canvas-apron frame.

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8 - Purington, C. W., Methods and Costs of Gravel and Placer Mining in Alaska: U. S. Geol. Survey Bull. 263, 1905. 273 pp.

9 - Storms, W. H., How to Make a Rocker: Eng. and Min Jour., June 24, 1911, p. 1243.

E. Cleats for holding brace at top of rocker.

F. Cleat for holding sieve box.

X. Bolt holes for $\frac{1}{2}$-inch iron tie bolts.

I. Riffles, $\frac{3}{4}$-inch high by 1 inch wide.

K. Rockers.

H. Handle for rocking.

L. Bottom board of rocker which should be in one piece or of matched-board construction $\frac{3}{4}$-inch thick.

M. Spikes projecting $1\frac{1}{2}$ inches to prevent rocker from slipping down grade.

The sieve box should fit loosely in the top of the rocker. The sieve is made of heavy sheet iron perforated with holes of about $\frac{1}{2}$-inch diameter.

The apron frame is of 1 by $1\frac{1}{2}$ inch material well fitted together and covered loosely with canvas so as to sag as shown, thus giving a slight depression in which gold is caught.

The rockers K rest on heavy planks bored to receive the spikes M. These planks are laid transversely, leveled up and set so that the one at the left in the illustration is about 2 inches lower than the other, giving a drop of 2 inches in 3 feet. This grade should be adjusted, however, to suit the character of material handled. If very little clay is present and the gold not excessively fine, the grade can be increased. The grade must be adjusted so that all the clay is thoroughly broken up before it is discharged from the rocker. If the gold is fine it may be advisable to add one or more riffles.

Operation of Rocker.- In operation the screen box is placed on the cleats F, and the gravel is shoveled into the box. The rocker is vigorously shaken back and forth with a jerky motion, while water is being poured over the contents of the box in quantities and at a rate which will thoroughly break up and remove lumps of clay and wash the gravel clean, yet not so rapidly as to carry small gold particles over the riffles. The flow should be regulated to just carry the tailings over the riffles and it is preferable to keep the flow fairly steady rather than in waves or surges. The water may be dipped up and poured over the contents of the box or be conveyed to the rocker in a small stream by means of a pipe or box. The latter means when available will obviously be less laborious. When heavy sands build up behind the riffles to the level of the top of the riffle, gold particles are apt to be washed over and lost. It is therefore necessary to keep an eye on the riffles and clean up the sands before they build up too high. If operated along a stream the question of water supply is obviously simple. If water is scarce or the placer is some distance from a stream, water for washing must be conserved. For this purpose pits may be dug at the head and foot of the rocker and connected by a small ditch, the water being reused over and over again. Four to six barrels of water per day are required ordinarily.

After the sand and clay have been washed away and the water coming through the screen is clear, the contents of the box, consisting of pebbles too large to pass through the perforation, are examined and picked over for any large nuggets that may be present, and are then discarded. The box is refilled and the operation repeated.

Clean-up.- The canvas apron is removed from the rocker and rinsed off in a tub of water several times a day, and the sands behind the riffles are cleaned out as often as may appear necessary. The concentrates from the apron and from the riffles are cleaned up in a gold pan as described under "Panning." Mercury is sometimes added to the riffles to catch fine gold.

The rocker is not very efficient and may lose a good deal of gold, yet it will handle considerably more material than a pan. It is employed for mining small patches of rich placer ground, in connection with exploration work and sampling placer ground, and has been used to some extent in working beach placers. It can be used in regions where the water supply is limited, provided the available water is carefully conserved.

Long-Tom

The long-tom consists of an open box 6 to 12 feet long (fig. 20) with a perforated plate or screen S at the bottom, into which the gravel and running water are introduced by means of a flume or box-launder F. The material passing through the screen openings which are usually about ½ inch in diameter, drops onto a set of riffles placed in another box R. The boxes are set on a slope varying from about 1 : 12, to 1.5 : 12. The amount of gravel which can be treated per day will vary with its nature, the water supply and the number of men employed to shovel into the tom and to fork out the large stones. Wilson[10] states that 2 men, 1 shoveling into the tom and the other working on it, can wash 6 cubic yards of ordinary gravel or 3 to 4 cubic yards of cemented gravel in 10 hours. At times the tom is operated by 4 men, 2 shoveling in, 1 forking out stones, and 1 shoveling fine tailings away. Toms are now rarely used in the United States; where running water and grade are available, a simple sluice is as effective and requires less labor.

Operation of Long-Tom.- The gravel is shoveled into the tom or flume, where the fine material is washed through the screen and the larger rocks are forked out and discarded. The gold and heavy sands are caught behind the riffles. When the riffles become filled up, the material caught behind them is removed and cleaned up in a gold pan. Mercury is sometimes added to the riffle box to catch fine gold. Figure 21 shows several common types of riffles and the method of placing them in boxes or sluices.

For long-tom operation a good supply of running water is essential. The drops between the boxes serve to aid in breaking up lumps of clay and to free any gold particles locked up therein.

A modified form of long-tom has been employed for washing beach sands at Nome, Lituya Bay, Yakataga, and Kodiak, Alaska. For saving fine gold the box is set at a high gradient, 3 to 4 inches per foot, and the screened material is passed over riffles and amalgam plates.[11] The water for sluicing is generally bailed with a large dipper.

Surf washers.- Surf washers are similar to long-toms but are wider and shorter.[12] They can be used only when the surf is of proper height. They are set so that the incoming surf rushes up the sluices, washes material from the screen box or hopper, and on retreating carries it over the riffles and plates. The average duty per man per 10 hours for long-tom or rocker work on beach operations is 3 to 5 cubic yards. One man can attend to two surf washers, and in one instance 8 cubic yards per 10 hours were handled.

Sluicing

Sluicing is a method capable of a number of variations and of adaptation to a variety of conditions; however, it depends for its success upon a plentiful supply of water and is most efficiently and easily conducted where the bedrock has a good natural gradient or slope. Table 1, taken from Peele's Mining Engineers' Handbook shows the principal variations of sluicing, including both hand and power methods.

In handwork a favorable natural slope is required and where natural slope is not available it becomes necessary to adopt mechanical means of handling the material in order to operate efficiently.

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10 - Wilson E. B., Hydraulic and Placer Mining: 3rd ed., John Wiley and Sons (Inc.), New Yok, 1918, 425 pp.

11 - Wimmler, Norman L., Placer Mining Methods and Costs in Alaska: Bull. 259, Bureau of Mines, 1927, 236 pp.

12- Wimmler, Norman L., Work cited.

Table 1.- Variations of sluicing and hydraulicking

Method of excavation	Gravel loaded into	Method of transporting gravel to sluice	Remarks
Running water aided by picking	Ground sluice	Gravity	Called "ground sluicing."
Pick-and-shovel	Sluice	Shoveling	Called "shoveling in."
	Wheelbarrows or cars on tracks	Hand-tramming	-
	Cars on tracks	Hand-tram to an incline and hoist to sluice on surface	Used where bedrock and topography are flat.
	Buckets or stone skips	Hand-tram on trucks to derrick or cableway, which hoists to sluice on surface	
Plow and scraper	Scraper	Scraper	-
Power scraper	Scraper	Scraper	Scraper may be hoisted up an incline to an elevated sluice.
Steam shovel	Cars on tracks	Animal or mechanical haulage	-
	Washing plant	Steam-shovel dipper	-
Drag-line excavator	Scraper bucket	Scraper bucket	-
Water under pressure	Ground sluice	Running water	Called "hydraulicking."
Mechanical excavator mounted on a boat, with screens, sluices, etc.			Called "dredging."

Ground Sluicing.- In ground sluicing, a stream or a portion of it is diverted to flow against or over a bank of placer ground, eroding it away and washing it to and through box sluices. The most favorable conditions for ground sluicing are usually found on the benches and upper reaches of the creeks. Ground sluicing is particularly useful in stripping light overburden from the pay gravel, and deposits of considerable thickness may be handled in this manner if conditions as to volume of water, character of the gravel, and the bedrock slope are favorable. It requires about six times as much water for ground sluicing as it does to do the same work in a box sluice.

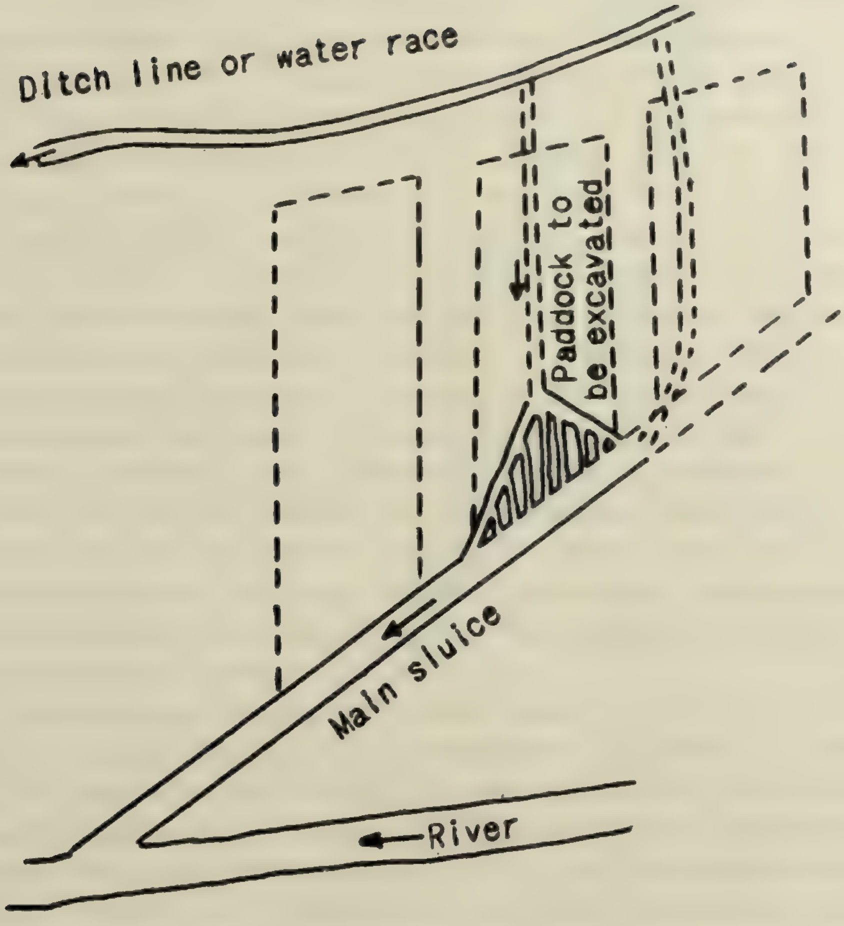

Figure 22.- Layout for ground sluicing (after Thorne and Hooke)

In some instances the stream may be diverted to run against a bank so as to undermine it and cut it away. In other instances a portion of the stream may be conducted from an up-stream point, by means of a ditch or flume, to the high side of the deposit, where it is released through a cross ditch or series of ditches in order to cut away the material and wash it along to the sluices (fig. 22).

A considerably steeper grade is required to handle gravel than to handle light overburden. Commonly ground sluicing serves to remove only the lighter material, leaving behind the gravel and pay dirt. It then becomes necessary to treat the remaining material in sluices by "shoveling-in" or handling it in cars, or by scrapers or other mechanical means. Scrapers may be drawn by horses or by hoisting engines. Where ground sluicing is to be followed by shoveling-in, the water is usually first diverted through a single cut, commonly 12 to 16 feet wide, along one side of the pay channel. The overburden and as much of the gravel as it is practicable to remove are sluiced through this cut to concentrate the gold in the remaining gravel. Bowlders are stacked to one side. Often there will be 1 to 4 feet of gravel remaining, which is then shoveled into sluices. Where conditions are such that it is possible to ground-sluice to bedrock, and where the bedrock is rough and thus affords a natural riffle, the gold and heavy sands may be found concentrated in depressions in the surface from which it may be cleaned up by panning or rocking. Comparatively steep grades and a plentiful water supply are necessary for ground sluicing. Sometimes spring floods can be utilized to remove much of the lighter material by ground sluicing, the remaining material being sluiced by shoveling-in during the low-water seasons. Longridge[13] states that two men can move a maximum of 20 to 30 cubic yards per day by ground sluicing.

Booming.- Dams are usually constructed for diversion of water for sluicing operations. Booming is a variation of ground sluicing in which the water is accumulated behind the dam and is released at intervals by hand or by mechanically operated gates; the water rushes or "booms" down the cut, carrying the material with it.

Shoveling-in.- Shoveling-in is adapted to rich placer gravel up to 6 or 8 feet in depth, or to gravel which has already been partly concentrated by ground sluicing or booming. It consists of hand-shoveling the material into sluices, cleaning up the bedrock, picking out the bowlders and stacking them to one side, disposing of the fine tailings from the sluices, cleaning up the riffles, and recovering the gold from the riffle concentrates.

A sluice is simply a series of telescoping open boxes (fig. 23) in sections usually 12 feet long and provided with riffles. Favorable conditions include a bedrock slope as steep as or, preferably, steeper than the desired grade of the sluices, plenty of room for disposal of tailings, sufficient water at all times, and a surface and bed rock contour which can be drained so that the shovelers will not be working in water.

Sluices are usually set on a grade of 6 inches to the 12-foot box or sluice section (4.16 per cent). Sometimes a long string of tailing boxes without riffles are required for disposal of fine tailings. In other instances bedrock grades are not steep enough and it becomes necessary to elevate the sluices on a trestle to attain the required gradient for sluicing. The maximum practical lift for shoveling into elevated sluices is 6 to 8 feet. The amount of gravel which a man can shovel in varies widely, depending upon the height of lift, amount of picking required to loosen the gravel, difficulty in cleaning bedrock, and the quantity of large bowlders to be handled. Only the lighter material is shoveled into the sluices; the rocks and bowlders are cleaned off and thrown aside on bedrock which has been cleaned up. Wimmler[14] states that in Alaska one man will shovel from 2½ to as much as 10 cubic yards in 10 hours with lifts of 3 to 7 feet; the depth of material shoveled is usually 1 to 5 feet. An average day's work is 7 to 8 cubic yards.

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13 - Longridge, C. C., Hydraulic Mining: Published by Mining Journal, London, 1910, 352 pp.

14 - Wimmler, Norman L., Work cited.

Riffles are constructed in various ways, of which some of the most common types are illustrated in Figure 21. Pole riffles are popular in hand-sluicing because they may be constructed of material usually available nearby for the cutting. The riffles are fitted and wedged in, rather than permanently secured, since it is necessary to remove them for the clean-up. For fine sand and gravel, smaller riffles or carpet, blanket, burlap, or cocoa matting protected by expanded metal or riffles are sometimes used on the bottom of the sluices to collect fine gold. Fine material containing fine gold usually requires shallow, wide sluices set on comparatively steep grades. Coarse material requires a narrow, deep sluice. If there is much fine gold, mercury is sometimes used in some sections of the sluice.

Water is supplied to the head of the sluice by a pipe or flume, and about 1.3 to 3 cubic yards per 24 hours can be handled with a flow of 1 cubic foot of water per minute. This is equivalent to a total of about 3,600 to 8,300 gallons of water per cubic yard handled.

The following table[15] gives two examples of sluice capacities.

Width of sluice, inches	Depth of flow, inches	Grade, per cent	Water flow per minute, cubic feet	Gravel washed per 24 hours, cubic yards
10 - 12	6 to 7	4.16	45	67.5 to 135
12 - 14	10	6.2	100	150 to 300

A grizzly or heavy screen box is sometimes placed in the head or upper sluice box onto which the gravel is shoveled. Thus the coarse material can be eliminated from the sluice at this point.

Clean-up.- Sluices are usually cleaned up at more or less regular intervals. Clear water is first run through the sluice until it is free from gravel. Then beginning at the upper end, the riffles are removed section by section, the while a light stream of water is running in the sluice which washes the lighter material to the sections below. The gold, heavy sands, and amalgam (if mercury has been used) are scraped up and placed in buckets. This material is then cleaned up in a rocker or pan in the usual manner. Sometimes the heavy sands are amalgamated after removing the visible gold, to recover any gold remaining therein.

Variations of shoveling-in.- Shoveling-in may be done by moving the sluice line to keep it close to the gravel bank and as the bank is cut away, moving the sluice over accordingly. In other instances where this would be expensive and result in frequent delays while moving, or where the slope of bedrock would not permit of this practice, the sluice is constructed in a semipermanent manner and located centrally with respect to the area to be worked; the excavated gravel is brought to the sluice by wheelbarrows or in cars running on light rails (fig. 24). In order to obtain sufficient grade for the sluice boxes and at the same time to have storage room for the tailings, it may be necessary to elevate the sluice to a height too great for shoveling-in. In this event it will be necessary to load the material into a skip at the foot of an incline and pull it up to the head box with a small hoist, or the cars themselves may be pulled up in this manner.

Undercurrents.- Undercurrents, which are a form of sluice but are shallow and are 8 to 10 times as wide as the main sluice, are often employed to recover fine gold which is not caught in the sluice boxes. Undercurrents are divided into compartments for convenience in placing riffles and in cleaning up, and to allow better control of the distribution of the

15 - Young, George J., Work cited.

material. These boxes are paved with wooden blocks, cobbles, or pole riffles shod with iron; the grades required for these riffles are 14, 16, and 12 inches per 12-foot box, respectively. Sometimes they are covered with carpet, cocoa matting, or burlap to catch fine gold. Undercurrents are placed alongside and below the main sluice at a point where the latter is sufficiently elevated to give the required slope from the main sluice to the undercurrent. At this point a grizzly or plate screen is placed in the bottom of the sluice. The coarser material passes over the screen while the fine material is washed through into a short launder or trough in which it flows to the head of the undercurrent.

Mercury in riffles.- Mercury is often used in riffles and undercurrents. Amalgam is easier to handle in clean-ups than is fine gold, and some gold which would otherwise be lost is usually saved by amalgamation. Mercury is not used in the first few sluice boxes, where most of the coarse gold is caught. Ordinarily 80 per cent of the total gold recovered is saved in the first 200 feet of sluice boxes in California mines.

Power Methods

When the placers are covered by a thick overburden or are of very low grade, or where the slope of the ground or bedrock is not suited to simple hand-sluicing methods, it becomes necessary to employ mechanical or hydraulic equipment for successful low-cost operation.

Drag Scrapers.- One of the simplest methods of excavating and handling gravel is by means of a drag scraper operated by a double-drum hoisting engine. The hoist is located at the head of the sluice, and, by an arrangement of snatch blocks and movable tail sheaves, the ground can be excavated and moved from a considerable area at one set-up of the hoist (fig. 25). Cableway excavations may be made in a similar manner.

Hydraulic Mining.- One of the simplest and cheapest methods of excavating and transporting placer material to the sluices is by hydraulic mining, provided conditions are suited to the use of this method. The term "hydraulic mining" is applied to the method whereby water under high pressure is employed for breaking down the bank and disintegrating the gravel. The water is conveyed by pressure pipe lines to nozzles, called "giants," by means of which the water is directed against the gravel bank to break it down and wash it to the sluices. Where the hydraulic water is not of sufficient volume to wash the disintegrated material to the sluices, the hydraulic water may be supplemented by running water for this purpose.

Hydraulic mining requires an ample supply of water under high head, a good natural bedrock slope, and ample room for disposal of tailings from the sluices. Hydraulicking is usually applied to mining thick deposits or gravels covered by thick overburden, but in Alaska it has been employed for mining creek placers and bench deposits, many of which are only slightly above the creek level.

The grade of the bedrock should be at least 2 per cent for finely divided material and 4 to 5 per cent for medium-sized material. In places where there is insufficient elevation for moving the gravel to and through the sluices and for disposal of the tailings, it may be necessary to wash the material to a pit or sump excavated in the bedrock, whence it can be elevated by means of hydraulic elevators to sluices supported on trestles. Hydraulic elevators also require the use of water under pressure. A head of 5 to 10 feet on the hydraulic water is required per foot of elevator lift. The efficiency of the elevator varies between 10 and [illegible]0 per cent.

At the Logan mine near Waldo, Ore., 15,000 to 30,000 cubic yards of gravel were washed per month with 40 cubic feet of water per second. Four giants were used, two in the pit and two on the tailings dump for washing away the tailings. Twenty-inch hydraulic elevators, lifted the material 49 feet in two lifts. The gravel was easily washed, and operating expenses were reported as being only 3-7/8 cents per cubic yard under exceedingly favorable conditions.

The equipment for hydraulicking is simple, consisting of pipes, gates, "giants," and sluices, and, in some cases, of hydraulic elevators. The cost of the giants is not high, but the development of the water supply may be expensive. This development may require the construction of dams, and long pipe lines or ditches.

It is not within the scope of this paper to discuss hydraulic mining and other more elaborate methods of placer mining such as those involving the use of steam shovels and dredges, which require large capital outlay for exploration and equipment prior to productive operation.

Suffice it to say that some form of sluice is employed, often with elaborate undercurrents, for recovering the gold, and generally mercury is used in some sections of the riffles. Before passing to the sluices the gravel is sometimes further disintegrated and washed by being passed through trommels (revolving screens) upon which water is sprayed under pressure.

Dredging is usually employed for mining wide gravel deposits overlaid by fairly thick overburden where the surface and bedrock are flat. The excavating machinery and gold-recovering equipment are mounted on a large barge or float. The ground is excavated by a line of heavy buckets which deliver the excavated material to a hopper and thence to a trommel. The material then passes over sluices and undercurrents, where the gold is caught, and the tailings are stacked behind the dredge by a conveyor belt. The barge floats in a pond formed by filling the excavation made by the dredge with water. Water is added as required to keep the pond full and make up losses of water by evaporation and seepage. One dredge in California visited recently by the author was cutting out a width of 300 feet and was digging 56 feet below the water level, which was about 20 feet below the ground surface. The dredge buckets were of 18-cubic-foot capacity each. The dredge was operated by electric power with motors aggregating 1,200 hp. connected load. A special crew of 10 men cleaned up the sluices and undercurrents once a week.

Dry Placers

Placer deposits are found in some arid regions where ancient streams have dried up, or present streams only furnish small supplies of water at infrequent intervals.

Attempts to work these deposits by dry methods have not as a rule been attended by much success, though in some instances individuals have made wages by working with small hand-blowing machines.

In one more serious attempt to work a dry placer a Stebbins-Quinner machine was employed.[16] The gravel was dug with a steam shovel, pulverized and screened by the machine, and then passed over Stebbins tables which used air instead of water as a means of concentration. No successful large-scale attempt is known to have been made in this country by this method.

Sometimes placer dirt has been concentrated by winnowing[17] - tossing it in a pan until the lighter particles have been blown away, and finishing by mouth blowing. It is, of course, a wasteful method of concentration. In west Australia the conditions were favorable for dry-blowing, as the winds were strong and constant and the air was hot and dry. The method used there is slowly to empty a panfull of dirt into an empty pan placed on the ground. This operation is repeated again and again, and is followed by tossing in a pan, by "panning" as though water were being used, and finally by mouth blowing. The larger pieces of barren material are removed by hand.

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16 - U. S. Bureau of Mines, Placer-Mining Methods: Rept. of Investigations 2315, 1922, 4 pp.

17 - Rose, Sir T.K., The Metallurgy of Gold: 6th ed., Charles Griffin and Co. (Ltd.), London, 1915, p. 111.

Among machines used, the simplest consists of flat screens supported on a frame and shaken by hand, the material falling through and being winnowed by the wind. In other contrivances a bellows is added, worked by hand. In some machines the blast of air is used to keep the sand partly in suspension while it is moved by gravity down an inclined table which is furnished with riffles. The dry material is first passed through screens and then falls on an inclined canvas deck supported on another screen. Sudden puffs or pulsations of air furnished by bellows pass through the deck from below, throwing the sand up and allowing it to settle back again alternately; as a result the light material works down the table, while the gold is retained by the riffles, being too heavy to be tossed over them by the air.

RESUME

The foregoing notes are admittedly sketchy, but as previously pointed out, they are designed to answer the principal elementary questions embodied in the inquiries being received by the Bureau of Mines.

A vast amount of detailed information on placer mining may be found in textbooks and technical publications. A selected bibliography on placer mining is appended hereto for those interested in detailed information on this subject.

It may be well to reiterate in closing that the maps included with this paper show the principal locations in the United States from which placer gold is known to have been mined, but it should not be inferred that profitable operations can now be conducted at these places. At some of the localities shown placer gold is still being produced.

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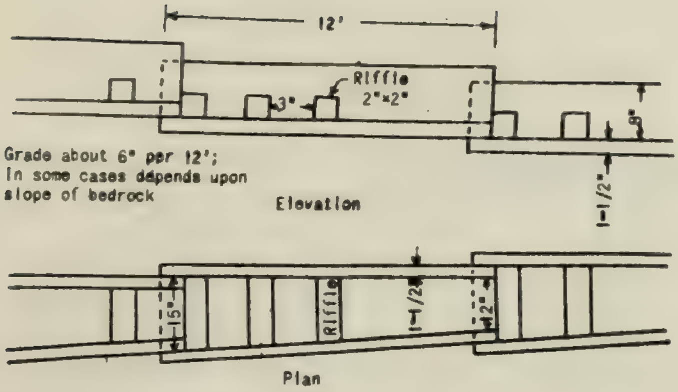

Figure 23.- One type of sluice construction (from W. W. Staley)

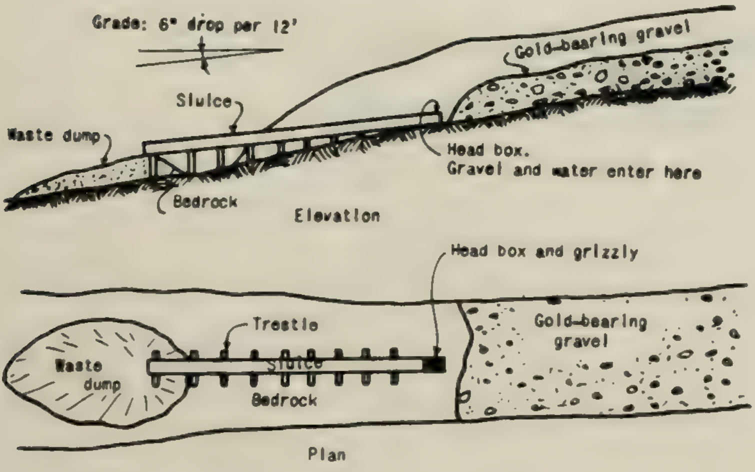

Figure 24.- Sluice layout for handwork (from W.W.Staley)

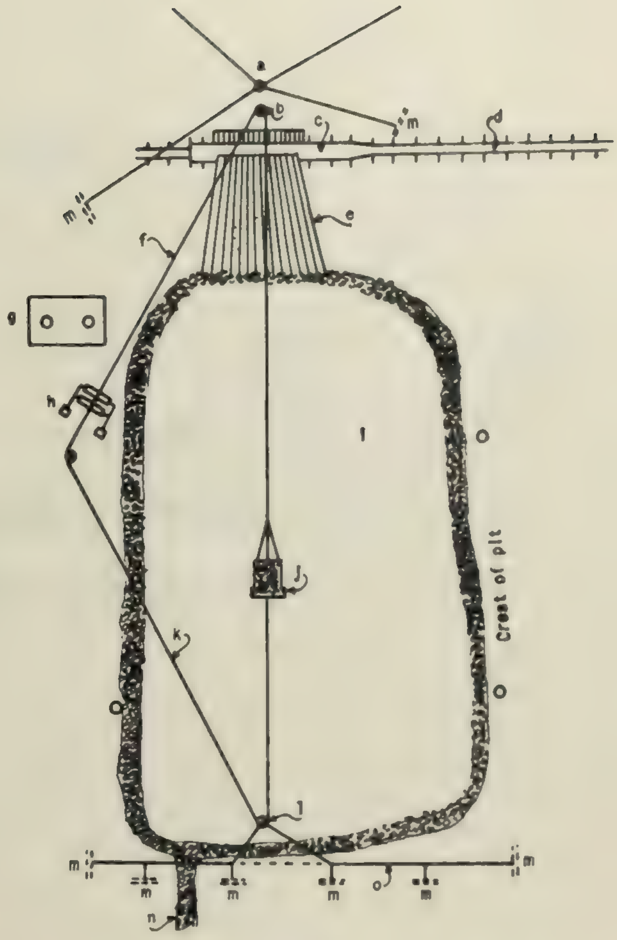

Figure 25.- Typical slip-scraper arrangement (after Wimmler): a, Gin pole; b, lead sheave; c, dump box; d, sluice; e, timber incline; f, lead cable; g, boiler house; h, 2-drum hoist, 8 1/4×10 1/4 inches; i, pit, 150×250 feet; j, slip scraper, 3/4 cubic yard; k, haulback cable; l, haulback sheave; m, deadmen; n, bedrock drain; o, 1-inch anchor cable

I. C. 6612 April, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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INFORMATION CIRCULAR

GOLD MINING AND MILLING METHODS AND COSTS AT THE VALLECITO WESTERN DRIFT MINE, ANGELS CAMP, CALIF.

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

GOLD MINING AND MILLING METHODS AND COSTS AT THE VALLECITO WESTERN DRIFT MINE, ANGELS CAMP, CALIF.[1]

By Don Steffa[2]

INTRODUCTION

Drift mining, according to California mining terminology - and it is distinctly a California development - is the blocking out and extraction (breasting) of the auriferous gravels of an ancient buried stream either through tunnels, inclines, or shafts; it is one of the minor divisions of the larger classification known as placer mining. The contribution of each of these divisions and of lode mining to the gross production of gold in California totals as follows:[3]

Type of mine	Period	Gold recovery
Open placer and ground sluicing ..	1900-1928	$ 8,791,208
Dredging	1891-1928	159,957,684
Drift mining	1900-1928	11,096,519
Hydraulic mining	1900-1928	14,668,631
Lode mining	1851-1928	596,646,956

The total amount of gold recovered from the placers of California from 1848, when Marshall discovered the precious metal in the mill race at Coloma, to 1928, was $1,226,915,404. The same period saw a total gold production from all operations of $1,823,562,360.

HISTORY

Drift mining in California dates back to the State's earliest days of placer operations. Successive stages of previous metal extraction from gravels ran the gamut from an ordinary water bucket filled with "pay dirt," which was sluiced around until the contents could be thrown on the ground and the gold picked out, to the cradle and rocker, to ground sluicing, and thence to the Long Tom. Hydraulicking entered the arena in the early fifties.

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1 The U. S. Bureau of Mines will welcome reprinting of this article provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6612."

2 Consulting engineer, U. S. Bureau of Mines, and general manager, Vallecito Mining Co.

3 Hill, J. M., Historical Summary of Gold, Silver, Copper, Lead, and Zinc Produced in California, 1848 to 1926: Econ. Paper 3, Bureau of Mines, 1929, 22 pp. U. S. Bureau of Mines, Mineral Resources of the United States, 1928; Part I, pp. 286-288.

With the discovery of ancient, auriferous river beds, buried beneath a heavy overburden of volcanics, primitive placer methods were superseded in time by milling processes to disintegrate the gold bearing gravels. Much drifting was done, however, long before the advent of milling machinery. Prior to the latter's introduction, an ordinary long sluice carrying riffles was the sole device in use. The usual custom was to permit the partly cemented gravel, or that consolidated with heavy clay, to slack on the surface for some time before being shoveled into the boxes to be washed. At a later date, stamps were used in some instances where the gravels were tightly cemented, generally by iron solutions, which made crushing necessary. For the most part, however, some form of circular, revolving tube, into which the pay gravels are introduced to be broken up, washed, and freed of their oversize, has held sway from an early date up to and including the present time. From the beginning the gold has been recovered in long sluices with a series of Hungarian riffles in the head boxes and various forms of pole riffles below. Undercurrents, screen tables, jigs, concentrating tables, etc., for the recovery of the fines are all of a later date. The early drift miner knew none of these; hence the high values subsequently recovered from many old tailing dumps.

The shallow placers in the gulches and ravines and flowing rivers were the first, of course, to attract the attention of the initial influx of "miners." These surface "diggings" occupied their energies for a period of years. Extraction of values was comparatively easy; hence little equipment was needed. But even before the shallow placers were exhausted and through accident or cursory investigation, discovery was made of the ancient buried streams, in many instances the feeders of exceedingly rich surface gulches, gravel mining necessarily changed and drift mining methods came into prominence.

So far as known, history does not record the first authentic shipment or production of gold from a drift mine in California. There is evidence, however, from a journal of current events kept by B. F. Marshall who lived at Murphy, Calaveras County, that the first production of gold from a drift mine in the State was from Central Hill and Murphy's Flat in that county. The journal records that 20 pounds of gold was received at Murphy from Central Hill on February 18, 1850; other notations over a two-year period show other shipments. Gold from this area was gained by drift mining. The writer is indebted to Mr. Marshall's son, F. B. Marshall of Standard, Calif., for the use of the record book.

Central Hill is a mile-long segment of the north tributary of the Tertiary Calaveras, or Central Hill channel, as it is sometimes designated. The extreme west end of that portion of the dead river lies within three-quarters of a mile of the town of Murphy (fig. 1).

In the early years of drift mining activities in eastern Calaveras County were confined to that segment of the Central Hill channel southeast of Murphy, thence on down stream to the Douglas Flat area where many shafts were sunk along the ancient stream course. An excessive flow of water in the latter

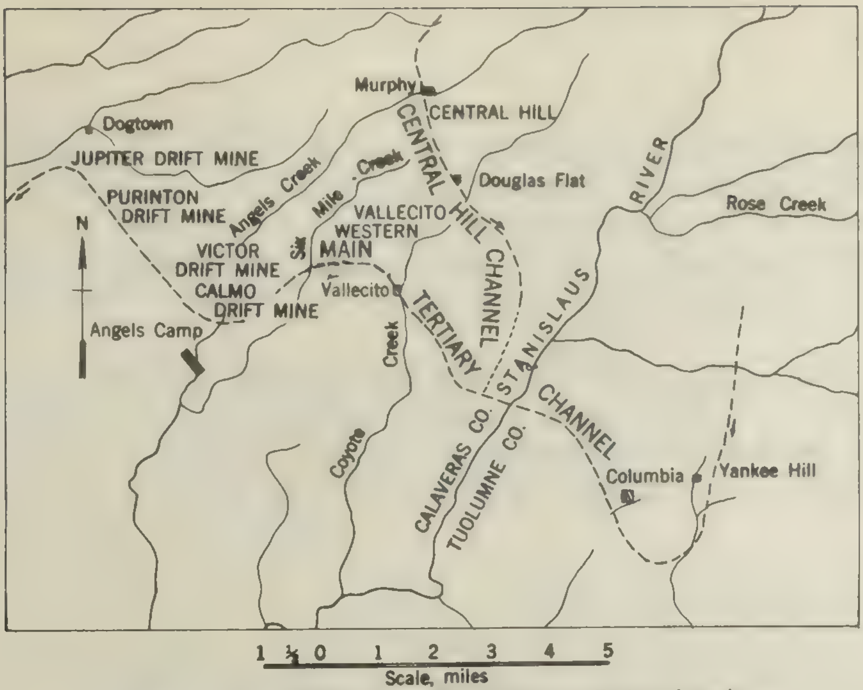


Figure 1. —Map of vicinity of Vallecito, showing main Tertiary channels

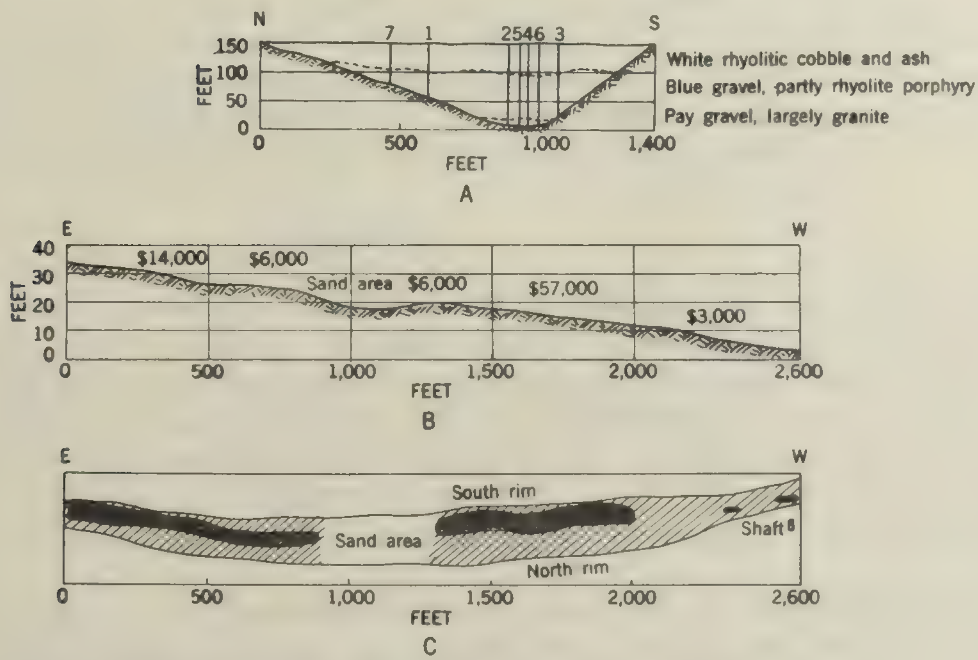


Figure 2.—A, Cross section of Tertiary valley at Six Mile Creek, showing churn-drill holes, numbered in order of drilling; B, profile of channel floor of Vallecito Western mine, showing amount of gold extracted during development of the indicated sections; C, plan of channel floor, showing pay areas in black

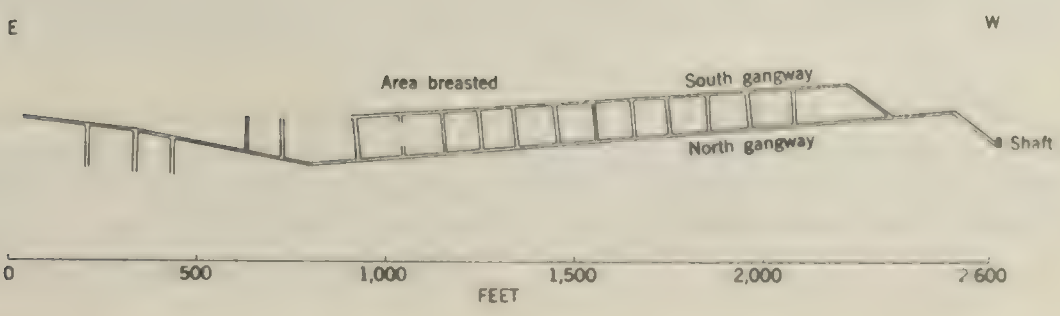


Figure 3.—Plan of principal workings, underground development, Vallecito Western drift mine

region, due in part to an accumulation in a fault plane paralleling the channel, brought operations to an early conclusion. Old-fashioned water wheels, 24 feet in diameter, were the only source of power at hand to drive the Cornish pumps, and these were inadequate to handle the flow. The Texas drift mine held the record in that section, continuing operations over a period of 17 years.

All the drift mines in the Murphy and Douglas Flat districts were operated through shafts, the contour of the country preventing the use of tunnels. Farther west, on the main tertiary stream, tunnels 500 feet long were driven in the early days to tap the channel at the Monarch, Bully Boy, Jupiter, and Purinton drift mines in the vicinity of Dogtown.

No work was done along the long stretch of the main ancient stream bed from Dogtown to Vallecito until comparatively recent years when the building of power lines through the area permitted the use of electric pumps. Water theretofore had proved a stumbling block to willing operators. In the vicinity of Angels Camp drifting operations began first at the Victor in 1905. Farther upstream the Calmo began sinking its shaft in 1922, followed at a point 2 miles farther east by the Vallecito Western, which company first drilled a cross section of the Tertiary valley to locate the channel and determine the exact depth through the rhyolite overburden to its floor. This work began in May, 1923, and was completed in the same month of the following year.

GEOLOGY

The Tertiary channel system in places in the vicinity of Vallecito (see fig. 1) is difficult to trace, owing to heavy flows of volcanic material and subsequent faulting and erosion which have left deep embayments and perplexing ridges of rhyolite, andesite, and latite. This is particularly true of the north tributary known as the Central Hill channel, which in part has been submerged under hundreds of feet of volcanic gravels. A portion of this channel has been affected by faulting having a vertical displacement of about 125 feet.

The headwaters of the main Tertiary channel were in the region near the present Rose Creek. The stream flowed southward for several miles and then made an abrupt turn to the northwest, passing the site of the present town of Columbia. At this point enormous enrichment of the gravels was found, due probably to the erosion of a heavily mineralized pocket belt in the low foothills near the town. Little overburden was present to hinder the mining of the rich gravels, and Columbia was one of the richest producing areas in the State. A total of $55,000,000 was mined between 1851 and 1871, and subsequent years witnessed the extraction of several million more.

The channel next crosses the mile-wide gorge of the Stanislaus River, passes Vallecito, and swings westward in a deep valley just north of that town. From this point, which is the site of the Vallecito Western operations, the stream is traced easily some 15 miles farther on to a point near Vally Springs, where the ancient river flowed into the sea.

The north branch of the Tertiary stream, known as the Central Hill channel, was followed southward by the early miners through Murphy to a point southeast of Douglas Flat, where it is dropped by a fault under a heavy overburden of rhyolite cobble and ash. Attempts to follow it by mining were prevented by heavy flows of water. Early geologic study indicated that it turned southwestward and joined the main channel north of Vallecito. The expenditure of much money and effort failed, however, to discover the channel in this stretch. Recent geologic investigation, aided by extensive underground development near Vallecito, has led to the conclusion that the Central Hill channel continued southward through Fairview and joined the main river, as indicated tentatively on the map shown in Figure 1.

From Vallecito to its mouth the ancient river channel is clearly defined by its intermittently outcropping bedrock rims, between which lie the flows of rhyolite cobble and ash. The latter are in places covered by later flows of andesite gravel. At Six Mile Creek the primary valley is 1,400 feet wide and the original depth of rhyolitic fill has been reduced by erosion to 153 feet, just east of the creek at the Vallecito Western shaft.

Exploration work for half a mile along the channel at the Vallecito Western showed that the channel proper is in places 275 feet wide, but that at no place did gravel of economic value exceed 80 feet in width. Figure 2 A shows a cross section of the ancient stream bed at Six Mile Creek. Figure 2 B is a profile of the channel floor at the Vallecito Western mine, showing also the amount of gold extracted from various portions of the development and blocking-out program. In the plan of the Tertiary channel at this point (fig. 2 C) the areas of economic grade are shown in black.

The present surface, where drilling was done by the Vallecito Mining Co., has suffered heavy erosion, the north-south canyon of Six Mile Creek having been cut several hundred feet below the rhyolitic capping which marks the Tertiary valley to the westward. As the course of Six Mile Creek might have been determined by faulting, the drilling results were studied to discover any considerable bedrock displacement, which would have been reflected by a break in the volcanic fill. However no faulting was disclosed.

Drilling attested the evenness of the stratification across the entire area prospected. The uppermost formation is composed of white rhyolite cobble and ash 40 to 50 feet thick, depending on the surface elevation. Flaky gold was found scattered irregularly throughout this formation. Immediately below this volcanic capping is a thick bed of well-rounded gravel deep blue in color, carrying a large quantity of porphyritic material. The later, however, is absent or rare near the bedrock, and the bottom pay gravels are largely of granite with a very small proportion of quartzite, jasper, and silicified slate. Quartz bowlders and gravel are almost wholly lacking. The gravels are compressed so tightly that blasting has been necessary both in drifting and in breasting, and little timbering has been needed. Yet the material is uncemented in that it is free from iron or siliceous binding which in other places has consolidated gravels so strongly as to require crushing by stamps. The bedrock is granite or a relatively soft slate.

The gravels of the Tertiary streams that drained Calaveras County received their enrichment largely from the quartz veins and stringers prevalent in the bedrock, augmented by the disintegration of the auriferous slates. The streams are barren where they reach back into the granite belt of the Sierras, the occurrence of gold being limited, broadly speaking, to the gravels derived from the metamorphic zone.

At the Vallecito Western mine, paying quantities of gold are distributed over long distances with remarkable uniformity and in many places to heights of 12 to 14 feet. In one drill hole, sunk in the trough of the ancient channel, pay gravels were found 23 feet above bedrock. In such areas, bedrock concentration is meager or lacking. On the other hand, in the upper end of the last body of pay gravel developed, there is a heavy concentration of values on bedrock and in crevices in the bedrock, whereas the gravels above were practically barren.

The gold recovered at the Vallecito Western has been of small size until recently. In the trial area breasted, where a total of $55,700 was recovered, there was found one nugget which weighed $21, and forty or fifty pieces ranging from $0.50 to $2.40. The balance consisted of fine particles so uniform in size as to appear to have been screened, the largest not exceeding $0.15 in value. In recent months, however, since opening the heavy bedrock concentration at the upper end, the character of the gold has changed from fine to coarse, and nuggets weighing as much as $30 each are taken from the riffles. The gold is clean, free from iron-manganese stains or sulphides. Only occasionally do pyrite-coated particles of gold appear in the clean-up. The gold produced to date has had an average fineness of 895, ranging from 882 to 900.

METHODS OF PROSPECTING AND DEVELOPMENT

Churn Drilling

The broad Tertiary valley east of Six Mile Creek was prospected by churn drilling to discover the location of the actual channel. A north-south line of holes was drilled across the valley, spaced as shown in Table 1 and Figure 1 A. The holes were not drilled in regular order across the valley. The first was drilled north of the valley center in order to determine the bedrock gradient of the north rim, and struck bedrock at a depth of 92 feet. This depth and the distance of the hole from the outcrop of the north rim of the channel permitted an approximate calculation of the slope of the north rim. Known elevations of the channel floor at points exposed by mining operations 2 miles west of this section indicated a depth of overburden here of about 150 feet. The second hole, drilled 281 feet to the southward, reached bedrock at a depth of 144 feet. The third hole was intended to be on the south rim and, in fact, struck bedrock at a depth of 135 feet. The fourth hole, with the information then available, was aimed for the channel trough, and the fifth and sixth were drilled to explore the width and possible pay area of channel gravels. The seventh hole was drilled north of the first hole to test for a possible split in the ancient river trough, but none was found, the hole bottoming on bedrock at 72 feet.

Table 1.- Drilling order, location, depth and size of churn-drill holes

Cross-section number	Order in which drilled	Location	Lateral interval, feet	Depth feet	Size, inches
1	7	North rim	---	72	6
2	1	North rim	135	92	6
3	2	North rim	281	144	12
4	5	North rim	43	142	6
5	4	Trough	30	145	6
6	6	South rim	31	142	6
7	3	South rim	67	135	6
---	8	Shaft site	---	187	12

The drill sludge from the first 100 feet of holes in the vicinity of the channel proper was sampled intermittently. From that depth to bedrock each 5 feet was sampled carefully. The sludge was weighed first and then passed through a rocker. The concentrate was panned and the gold weighed.

The drill rig used was an assembled outfit purchased by the company and was powered by a 10-hp. gasoline engine. The crew consisted of a runner and a helper. Six and twelve inch "Mother Hubbard" type bits were used.

All of the seven prospect holes were drilled without casing, except one where it was intended to sink a prospect shaft; the plan was later abandoned, however. Two more holes were lost, one at 82 feet due to casing trouble, and one at 57 feet when a large quartz bowlder was encountered which neither hammering nor blasting would dislodge. A tenth hole was sunk at the present shaft site, as described later. All drilling was done on company account. The total footage was 1,198 feet, drilled in 278 working days, or an average of 4.3 feet per day. The average cost was $6.11 per foot. The costs of drilling the 6-inch and 12-inch holes were practically the same.

As it happened, the estimates based on the results of this drilling and sampling were proved by later underground development to be fairly accurate. Drilling, however, should not be depended on for the exploration of such deposits, as the gold concentration is usually erratic. The recovery of gold from the drill sludge should be taken merely as an indication of the presence of ore in the stream gravels, the quantity and distribution of which must be determined more closely by drifting and finally by extraction.

Shaft Sinking

The shaft of the Vallecito Western was located at a point 50 feet north of the actual channel (see figs. 2C and 3) in order that the shaft station, at a depth of 153 feet below the collar, might be in the solid slate bedrock. At the point selected the shaft passed through 143 feet of volcanic cobble, ash, and sand and gravel before reaching the slate. It was sunk a total depth of 167 feet, providing a 14-foot sump below the station.

The shaft is 4 feet by $7\frac{1}{2}$ feet in the clear and has one 4 by $4\frac{1}{2}$ foot skip compartment and a $2\frac{1}{2}$ by 4 foot manway. It is timbered with 8 by 8 inch Douglas fir, excepting that 6 by 8 inch material was used for dividers, and is lined with 1 by 12 inch boards.

The shaft was sunk to bedrock without blasting, picks and gads being sufficient to loosen the material for shoveling. The 24 feet through rock was sunk by hand drilling, using 10 to 12 holes per round, light charges of powder, and electric delay detonators.

A 12-inch churn-drill hole was sunk first at one end of the shaft to handle the flow of water which was struck at a depth of 8 feet and amounted to about 35 gallons per minute throughout the work. The hole was sunk to a depth of 187 feet and cased with perforated 7-inch inside diameter stovepipe casing. A deep-well type of turbine pump was installed which was powered with a 20-hp. vertical electric motor, the motor resting on staging about 4 feet above the shaft collar. Three-foot lengths of pump column were used, and as the shaft deepened from day to day enough blocking was removed from under the motor support to keep the pump intake at the level of the bottom of the shaft. When blasting, during the latter part of the work, the casing and pump column, exposed in one end of the shaft, were protected from damage by a heavy plank hung from the bottom end-plate directly in front of the drill hole.

Numerous strata of sand and volcanic ash were encountered, one such bed at a depth of 70 feet being 7 feet thick. A large part of this fine material was carried to the surface by the pump. A test showed that at one time the pump discharge was one-third sand by volume. The pump impellers wore rapidly, three sets being used. Moreover, the drill hole rapidly filled with sand to the level of the pump, after which the pump could not be lowered farther. Twice the pump was removed and the hole cleaned with a sand pump. Finally, at a depth of 75 feet, this difficulty was remedied by cutting a slot in the casing of the hole, wide enough to insert a hand to clean out the sand from under the pump. As the shaft deepened, the slot was likewise cut down. To secure suction with a shallow sump, such as could be dug out easily by hand in this manner, a 4-inch strainer was substituted for the original 3-foot one. The pump was run continuously and regulated by the gate valve on the discharge pipe to the exact amount of water flowing into the small sump.

It required 90 days to complete the shaft. The average progress in sinking, including timbering, was slightly less than 1 foot per shift, working two 8-hour shifts per day. The cost was $39.50 per foot. Shaftmen and the foreman received $6 per day and engineers $5. Timber and lumber laid down at the shaft cost $42 per thousand board feet.

Drifting and Crosscutting

A diagonal crosscut was run southeast from the shaft to the north or near side of the channel, and a drift started upgrade (east) along the north rim, as shown in Figure 3. About 600 feet east of the shaft a crosscut was driven through the pay gravels to the south rim. Bedrock dropped away southward, and a winze at the south rim showed it to be 5½ feet lower on this side than on the north. This was due probably to the channel here being entirely in slate, which had permitted the cutting of a deep trough next to the abruptly rising south rim.

To the eastward a harder granite floor gradually encroached from the northward upon the channel until it covered its entire width, whereupon the bedrock assumed equal elevations on both sides. However, it was necessary to reach grade on the south side, and therefore a crosscut was started, as shown in the diagram, about 250 feet from the shaft. At this point the site of an ancient waterfall had been encountered with a rise of about 5 feet, and the grade of the north gangway had been raised correspondingly by the installation of a transfer platform. Loads coming down were dumped at this point through a hole in the platform into cars, which were then trammed to the shaft. The crosscut was therefore extended along the west or low side of the falls and at the south rim was turned east and driven in slate on a grade to intercept the bottom of the deep trough discovered at the first crosscut.

Crosscuts are run at intervals ranging from 100 to 150 feet. A total of 44 have been driven to date (August, 1931), 18 of which were in the first or westernmost of the pay areas developed. Of these, several connected the north and south drifts or gangways, serving both to improve ventilation and to speed up the work of breasting. The other crosscuts were projected away from the pay areas onto benches and were extended short distances up the rims to prospect for potential concentrations. The total footage of drift and crosscut to date is 6,300 feet.

Both gangways and crosscuts are generally 7 by 7 feet in section. The usual drill round consists of six holes drilled 5 or 6 feet deep and breaking an average of 4 feet per round. The gravel drills easily, 2½ hours generally being sufficient to drill the round. Drill steel is of 7/8-inch hollow-hexagonal material, sharpened with cross bits. Slightly more than 9 pounds of 25-per cent strength powder is used per round, with 4 sticks in each of three lifters, 3 sticks each in the two cut-holes, and 2 in the single back hole. Caps are treated with a standard waterproofing compound.

The broken gravel is shoveled by hand into 18-cubic foot end-dump roller-bearing cars holding 1 ton each. Track consists of 16-pound rails laid to 18-inch gage. The grade of the channel has proved uniform over considerable distances and averages 75 feet to the mile. Track has been laid therefore on a grade of 1½ per cent upstream. It has seldom been necessary to take up bed-rock to maintain the grade; wherever a dip in the floor has been found, the track has been kept on grade, and bedrock has always been found at the expected elevation when reaching the opposite side of the dip.

In the opening of new areas by drifts or crosscuts, samples are taken from the skip at the collar of the shaft, a sample consisting of one full pan or about 20 pounds of gravel. Samples taken at this point have the advantage as compared with samples taken from the solid face of being representative of a larger volume of ground and of being mixed thoroughly by the blasting and by the handling of the gravel from muck pile to car and to skip. Thus an experienced panner is able to make fairly accurate estimates of the value of the gravel developed.

Drifts and crosscuts are driven by crews of three or sometimes four men, making an average advance of 4 feet per shift. The cost of driving main headings averages \$16 to \$17 per foot. In a pay area 65 feet wide, where gravel can be breasted 10 feet high, each foot of heading develops 45 tons of gravel. (It is estimated that the gravel expands one quarter on being broken and a ton of broken gravel has a volume of 18 cubic feet.)

Breasting

Figure 3 indicates the location of the one area breasted so far. This averaged 65 feet wide, ranging from 60 to 80, and was 240 feet long. Near its center the pay gravels extended to a height of 14 feet and were extracted to that distance above the floor.

Breasting began at the upper end, the gravel being broken down along the side of a crosscut. Holes 6 feet deep spaced 4 feet apart in two rows, one at the top and the other at the bottom, were drilled across the face. Light explosive charges sufficed to make a clean 6-foot break and to loosen a foot or two more of ground to be picked down by hand. Heavy blasting is avoided because of the scattering effect on the fine gravel and its gold content.

The gravel is compacted so strongly that it stands without scaling over great widths with only occasional light stulls for support. The stulls are 8-inch round timbers set about 10 feet from the face, topped by headboards or caps, and wedged tight to withstand blasting. In the entire area breasted only 48 stulls were used. The roof is arched from a height of as much as 14 feet in the center to 7 to 10 feet at the sides, which increases its strength and tends to prevent sloughing.

As soon as the first slice is broken down along the crosscut, mucking begins. The gravel is shoveled by hand into the cars. Large bowlders, constituting about 30 per cent of the whole mass, are rolled back from the face and sometimes stacked up to the roof to furnish additional support. Very heavy bowlders, weighing from a few hundred pounds to several tons, are rare. Fully 80 per cent of the total weigh less than 100 pounds.

The top of the pay gravel is defined by a capping of coarse sand. Horizontally, the extent of breasting is controlled by pan sampling underground, the number of colors in a single pan indicating to an experienced gravel miner the approximate value of the ground. In places at this mine the pay lead is heavily concentrated and narrows to a width of 20 feet with barren ground on both sides. At others, as noted, the width of pay gravel is 80 feet. The width of face is varied accordingly.

The gravel is trammed by hand in 1-ton (18-cubic foot) cars to the shaft and dumped directly into a $1\frac{1}{2}$-ton skip. The skip raises the gravel to the top of the 80-foot headframe, where it is dumped into a 75-ton gravel bin.

The breasting operation was conducted to discover by a mill test the actual value of a given large mass of gravel, as well as to learn the north-south limits of the pay streak. Only one face was attacked, whereas in regular operation each crosscut would give a starting point for two faces. In full-scale operation, moreover, mechanical loading at the breasts and motor haulage should lower the cost of operation.

Breasting operations extended over a 10-month period, during which development work also was being pushed. The tonnage from breasting was segregated and treated separately, totalling 9,500 tons. Five men breasted and trammed 1,300 feet to the shaft, an average of 5 tons per man-shift. Powder consumption averaged $\frac{1}{2}$ pound per ton and the timber cost was $\frac{1}{2}$ cent per ton.

Drainage

The gravels are wet when first opened, but drain rapidly, and at present the flow of approximately 48,000 gallons per day is confined to bedrock, flowing between rails in the drifts. Drips from walls or roof are found only occasionally. At the shaft a vertical centrifugal pump, driven by a 10-hp. motor, is mounted in the manway about 10 feet above the station level. This pump has a capacity of 100 gallons per minute. It is controlled by a float switch and handles the regular mine drainage with about eight hours of pumping per day. A second turbine of double that horsepower and capacity is installed at the opposite side of the shaft ready for use in emergency. Power is taken from a Pacific Gas & Electric Co. line which passes 600 feet south of the shaft

MILLING METHODS

The gravel is treated in a plant near the collar of the shaft having a capacity of about 15 tons per hour (see fig. 4).

From the shaft bin the gravel is washed by water from a 2-inch line through the bin gate, into and through an 11-foot Hungarian-riffled sluice. From the lower end of this sluice the gravel discharges into the hopper of a 3 by 18 foot trommel, set at right angles to the line of the sluice. This trommel has two compartments, one for washing and disintegrating, the second for screening and further washing. The first, 8 feet long, is of unpierced steel, lined on the inside with 4-inch angle irons. As the trommel revolves these fins lift the gravel and cascade it to the bottom again, producing a crushing and disintegrating action similar to that in a ball mill. The trommel is set on a slope of $\frac{1}{2}$ inch per foot and is revolved at a speed of 28 r.p.m. by a 10-hp. motor.

The lower 4 feet of the trommel consists of two concentric screens, the inner perforated with $1\frac{1}{2}$ and the outer with 3/8 inch holes. All the material from the disintegrating section of the trommel passes by gravity onto the $1\frac{1}{2}$-inch screen. The oversize is discharged into a steel-lined sluice 60 feet long. It is forced through this by a stream of water from a 6-inch line and passes first over 6 feet of Hungarian riffles, then over 100 feet of pole riffles, and then to the waste dump. The pole riffles are constructed of longitudinally placed 8-pound steel rails.

The undersize of the $1\frac{1}{2}$-inch trommel drops onto the 3/8-inch outside screen, where it is washed by a stream of water from a 1-inch line. The washed oversize of this screen joins the discharge of the coarse screen. The minus 3/8-inch material drops to a 4-foot Hungarian riffle and passes over this onto a $3\frac{1}{2}$ by $6\frac{1}{2}$ foot "screen table." The latter is simply a $\frac{1}{4}$-inch mesh heavy-wire screen laid flat in a widened sluice box of the dimensions noted. Spreading the fines over this table permits most of the fine gold which has escaped the riffles above to settle in the interstices of the screen. No quicksilver is used. It has been found that under the conditions at this property the mercury "flowers" and gradually migrates under the action of swiftly flowing water. Fine gold which would otherwise remain in the riffles is then caught up and carried to the dump.

From the screen table the fines pass through a 20-foot section of Hungarian riffles, then over two 12 inch by 6 inch baffle plates, then over a final 12-foot Hungarian riffle. After this they join the coarse discharge of the trommel at the head of the pole riffles and pass eventually to the waste dump.

Above the baffle plates sluice boxes are 16 inches wide at the bottom and below the baffles are 12 inches wide. The function of the baffle plates is to smooth out the flow of sand and water, giving a more or less even feed to the last riffles. The first sluice box, between the bin and the trommel, is set at a slope of 2 inches per foot. The 4-foot segment between the trommel and screen table, the screen table itself, and the succeeding 20-foot section are all set on a $1\frac{1}{2}$-inch slope. Below the baffle plates the grade of the last section of Hungarians and of the pole riffles is $1\frac{1}{4}$ inches per foot. Any lower grades in the upper section of the plant result in crowding of the riffles with sand and overburden and the escape of fine gold due to the obstruction of effective water action.

Practically all of the coarse gold and 60 per cent of the total is recovered in the first 11-foot riffle immediately below the gravel bin. Following the washing and screening in the trommel, a very large part of the remainder is caught in the 4-foot Hungarian above the screen table. Occasional very large nuggets will accompany the oversize of the trommel and be caught in the first few feet of pole riffles.

When gold-bearing gravels are being milled, the upper two sluices are cleaned up twice a week, regardless of the tonnage, and daily if the gravels are unusually rich. The screen table is cleaned about once a month and the lower riffles only every six or eight months.

Clean-ups are started in the uppermost sluices. The riffles are lifted from the box, maintaining a reduced head of water, just sufficient to wash the sand and lighter material through the box as the mass is agitated slowly by hand and to precipitate the gold to the bottom. The last riffle remains in place and serves as a dam until the contents of the box have been reduced to a pan or less of concentrates. Coarse gold, if any, is screened out, and the fines are recovered on a small concentrating table. The percentage of flour gold is negligible.

The plant requires about 500 gallons of water per ton of gravel treated or in cubic measure about 3.7 cubic feet of water per cubic foot of gravel. The mine drainage water is discharged into a 400,000-gallon reservoir close to the shaft. From the reservoir the water is forced into a 1,500-gallon steel pressure tank by a 15-hp. automatically controlled centrifugal pump. When the air pressure in the top of the tank drops below 15 pounds per square inch the pump is started; when it reaches 40 pounds the pump is stopped. A pressure of about 25 pounds is maintained when operating the plant, and the pump runs continuously.

A second and higher reservoir of 4,000,000 gallons capacity has been constructed in a gulch 1,200 feet northeast. In the rainy season this serves to store run-off water, which can be drawn off as needed. It is dry during the summer months.

The centrifugal pump has a capacity of 500 gallons per minute, but only about 300 of this is needed for sluicing operations--100 in the upper line to carry the gravel through the trommel and 200 to carry away the oversize to the dump. The mill capacity varies from the 15 tons per hour noted above, with the character of the gravel, which ranges from well-rounded easily washed material to clayey or sandy gravel containing angular pieces of bedrock and flat pebbles.

The effectiveness of the gold recovery has been shown by a mill test of the tailing. Two hundred tons of these, rehandled, yielded $0.175 per ton. As the original gravel had averaged $8 per ton, the losses were only slightly over 2 per cent.

COSTS

The following costs are for combined extraction and milling of the 9,500 tons taken from the area breasted as described above. Prevailing wages during the ten months in question were $4.50 a day for muckers and trammers and $5 for miners. The costs, apportioned to the mining and treatment of the breasted gravels, excluding development but including all other operating costs of mining and milling, were as follows:

Cost of mining and milling

Labor	$2.02
Supervision and insurance	.40
Explosives	.25
Timber	.01
Power	.30
Other supplies	.12
Total, per ton	3.10

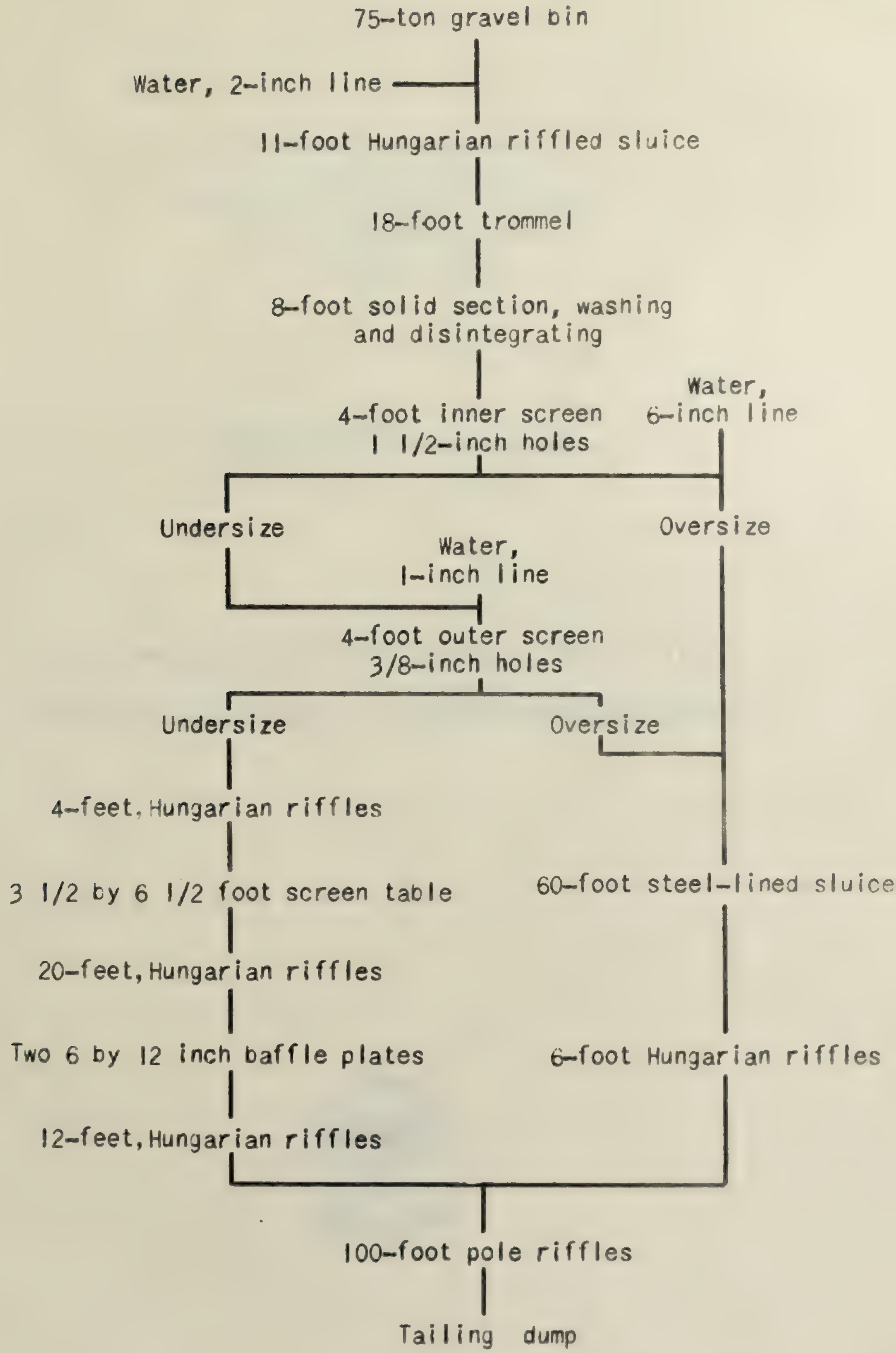
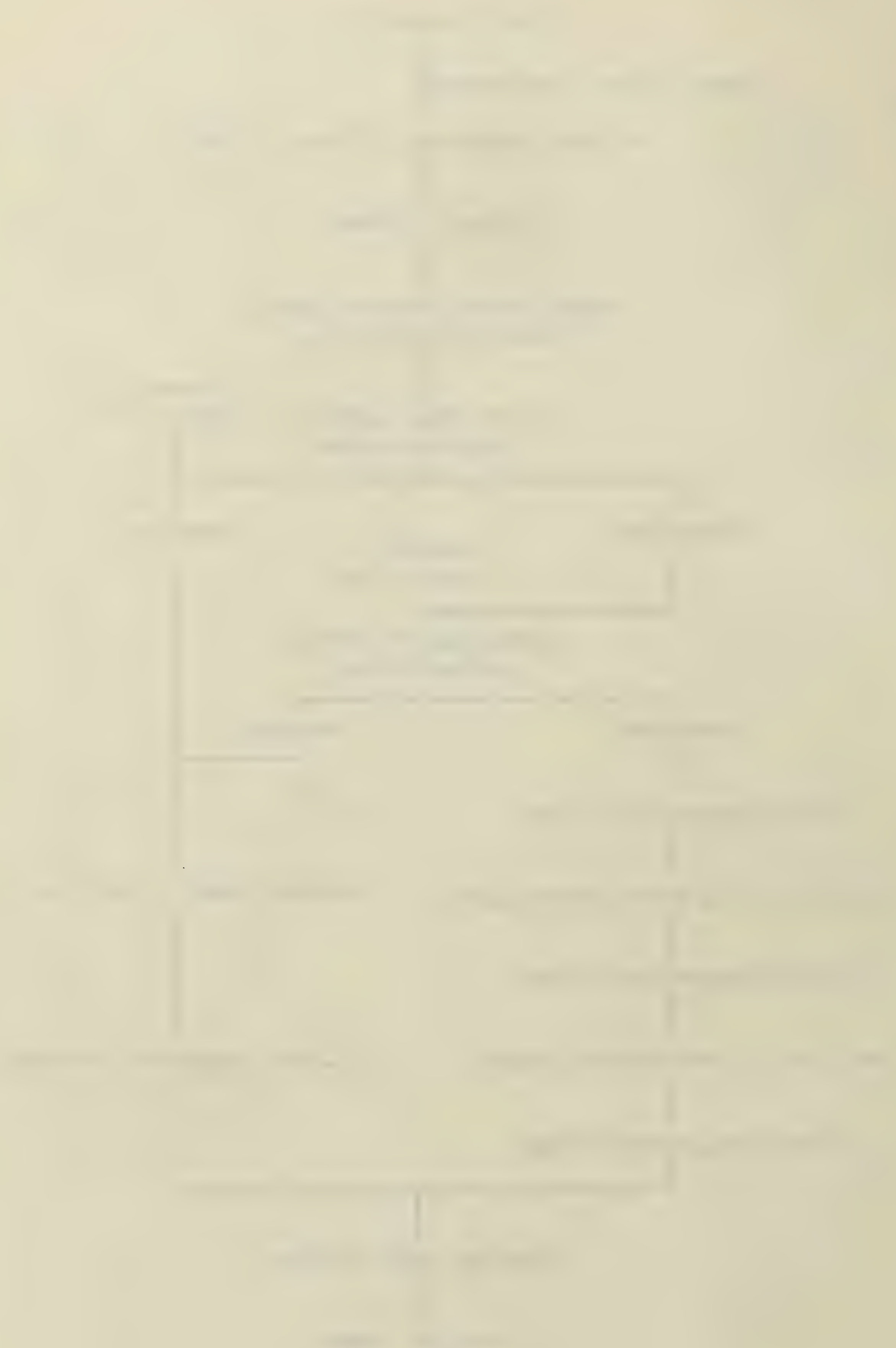


Figure 4.- Flow sheet of mill



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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

FACTORS GOVERNING THE SELECTION OF THE PROPER LEVEL INTERVAL IN UNDERGROUND MINES

BY

William O. Vanderburg

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

FACTORS GOVERNING THE SELECTION OF THE PROPER LEVEL INTERVAL IN UNDERGROUND MINES[1]

By William O. Vanderburg[2]

INTRODUCTION

Some mines after passing through the prospecting stage and becoming sizable enterprises still adhere to the development program of the prospect, extending their wo kings without definite plans. In mining operations where definite structural conditions do not exist, a prearranged layout of mine development is not feasible; but where exploration and preliminary development have indicated the extent and probable limits within which the ore is expected, careful planning of development will result in a lowering of mining costs and a consequent increase in profits. A lack of definite plans for laying out mine workings is one of the contributory factors to unnecessarily high costs. Although this paper deals primarily with the influence of level spacing on mining costs, still other phases of mine development, such as the proper spacing of ore chutes and raises, can be analyzed on a cost basis in a similar manner, as shown herein.

LEVEL SPACING

The problem of establishing a new level or levels in order to exploit steeply dipping ore deposits and prospect the ground at depth recurs at intervals during the active life of mines. Driving levels is largely "dead work" and therefore costly. By the judicious selection of the proper level interval, after balancing the opposing factors involved in the problem, a considerable saving in mining costs can be effected. Although much has been written in regard to the various methods employed in ore extraction, mining literature is still incomplete in regard to the factors influencing level spacing at various mines. Level spacing involves procedures not lending themselves readily to description, and, moreover, it is a phase of mining operations that is apt to escape criticism, because only those in close contact with the operations are equipped with the necessary familiarity with the conditions to pass judgment.

In former years when the bonanza orebodies were mined near the surface the necessity for close scrutiny of mining costs did not prevail to the extent that it does to-day, when the average grade of the ores mined is appreciably lower. Many mines yearly increase in depth, and mining costs tend to become higher because of the factors incident to deeper mining, so that the question of economical level spacing becomes increasingly important.

In many mines the distance between levels has been arbitrarily taken as 100 feet for no other apparent reason than that of precedent. Although the 100-foot spacing has proved economical in certain cases, it does not follow that the same spacing must necessarily prove so in others. According to Hoover,[3] "There was a time when mines were worked by driving the

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6613."

2 - Associate mining engineer, U. S. Bureau of Mines.

3 - Hoover, Herbert C., Principles of Mining: McGraw-Hill Book Co., New York, 1909, p. 88.

level on ore and enlarging it top and bottom as far as the ground would stand, then driving the next level 15 to 20 feet below, and repeating the operation. This interval gradually expanded, but for some reason 100 feet was for years assumed to be the proper distance between levels. Scattered over every mining camp on earth are thousands of mines opened on this empirical figure, without consideration of the reasons for it or for any other distance." It is worthy of note, however, that in deep mines (1,000 to 4,000 or more feet deep vertically) there is a tendency in recent years to increase the level interval (usually from the 100-foot spacing) due to the need for greater economy.

Although the selection of the most economical and advantageous spacing of levels for a particular mine rests fundamentally on factors which, in the majority of mines, can only be approximated, still by making compromises between the various requirements and conditions from a cost viewpoint, a sufficiently close determination can be made.

In Table 1 the average distribution of costs for the various underground methods have been compiled from data presented by Wright.[4] In Table 2 the same costs are expressed in percentages of the total direct costs. Attention is called to the fact that while development constitutes an average of only one-fifth of the total direct costs for all the methods of mining, the savings effected by the proper level spacing are reflected in the other costs as well.

TABLE 1.–Distribution of Direct Costs for Various Underground Methods of Mining

Mining method	Number of mines	Development	Mining	Haulage and hoisting	Other costs	Totals
Square set and fill	10	$0.928	$2.558	$0.657	$0.775	$4.918
Cut and fill	4	.707	1.672	.314	.516	3.209
Shrinkage	10	.617	1.406	.356	.152	2.531
Sublevel caving	2	.285	.379	.376	.398	1.438
Open stope	10	.267	.544	.376	.150	1.337
Top slicing	2	.152	.723	.179	.177	1.231
Block caving	4	.137	.174	.110	.095	.516

TABLE 2.–Distribution of Direct Costs Expressed in Percentages of Total Costs for Various Underground Methods of Mining

Mining method	Number of mines	Development, per cent	Mining, per cent	Haulage and hoisting, per cent	Other costs, per cent	Totals, per cent
Square set and fill	10	18.9	52.0	13.4	15.7	100
Cut and fill	4	22.0	52.1	9.8	16.1	100
Shrinkage	10	24.4	55.5	14.1	6.0	100
Sublevel caving	2	19.8	26.4	26.1	27.7	100
Open stope	10	20.0	40.7	28.1	11.2	100
Top slicing	2	12.3	58.7	14.6	14.4	100
Block caving	4	26.5	33.7	21.3	18.5	100
Weighted average of all methods	42	21.2	47.5	18.1	13.2	100

4 – Wright, Charles W., Mining Methods and Costs at Metal Mines in the United States: Inf. Cir. 6503, Bureau of Mines, 1931, 39 pp.

PURPOSES OF LEVELS

A level may be defined as the lateral workings (drifts and crosscuts) in a mine which are at the same or approximately the same elevation. In mining operations the term "level" is sometimes loosely employed to designate all the workings tributary to the level proper: namely, such workings as raises and stopes between two consecutive levels.

Levels are driven primarily for the following purposes: (1) To serve as a base for ore extraction operations; (2) to serve as passageways for workmen and the transportation of waste, ore, and supplies; (3) to serve as a datum for prospecting the contiguous ground. In the early stage of opening up a mine the use of a level for prospecting predominates, but after the mine has reached the productive stage the utilization of a level for prospecting is subordinated to its use as a base for the extraction of ore. Secondary purposes of a level are to afford drainage of the mine and the ventilation of the mine openings for the comfort and increased efficiency of the workmen. In a small mine a level may serve all the above-mentioned purposes at the same time, but in large-tonnage mines where transportation underground is an important item in mining costs, a portion of the level may be driven primarily as a haulage way, independently of the other purposes.

USE OF COST ESTIMATES TO DETERMINE THE PROPER LEVEL SPACING

It may be taken as axiomatic that the most desirable level interval in any underground mine is the one which involves the least possible expense per ton of ore extracted with minimum dilution compatible with safety and health of workmen and maximum recovery of ore.

In the selection of the proper level interval the problem centers largely on the estimated relative costs for establishing a level at various elevations. In order to compare the estimated costs they must be referred to a common datum and the only datum available is the probable ore developed per vertical foot of depth below the lowest level. Exception may be taken to basing cost estimates on a foundation which must necessarily be, in the majority of mines, an approximation, but it may be stated that the cost analysis does not require a great degree of refinement and any assumptions made are distributed equally for the various level spacings.

In large mines such as the porphyry coppers, the iron mines of the Lake Superior district, and certain mines in Canada, which are more or less delimited by means of drilling prior to development the problem can be brought to a greater degree of accuracy since more reliable data are available in regard to the extension of the ore in depth. In most small mines, however, exploration and prospecting are usually carried on concurrently with development so that the amount of ore developed per vertical foot of depth can only be approximated from geological conditions and the past history of the mine.

In a cost analysis to determine the proper level spacing a distinction must be made between expenditures which affect and those which do not affect the level interval. The latter items may be termed "capital" or "suspense" charges; they include the cost of such items as equipment for transportation, pumping, ventilation, and drilling. The value of such equipment is not wholly expendable during the life of any one level, and consequently these costs are distributed (theoretically) to the cost per ton of ore mined, either for the life of the mine or the life of such equipment. However, the cost of installing such equipment as track, air, and water lines, ventilation equipment, and the like must be repeated each time a new level is opened; therefore, such costs are directly chargeable to the ore mined from the particular level on which the equipment is used.

Shaft sinking is a capital expense charge and as such has no direct bearing on the problem of level spacing.

Mine workings driven solely for prospecting or exploration constitute capital expense charges. Where such workings are initially driven for prospecting or exploration and subsequently are employed for ore extraction, their costs then become directly chargeable to the life of the particular level they serve.

Certain items chargeable to the ore mined from a level may be constant for various level spacings in one mine, variable in another, or entirely absent in a third. Such an item is the cost of driving a crosscut from a shaft to an orebody; when the shaft and dip of the orebody are not parallel the cost of the crosscut is variable for different level spacings; and when the shaft and dip of the orebody are parallel the cost of the crosscut is constant for any level spacing. In the latter case, although the cost is uniformly the same per level, the total aggregate cost of such crosscuts during the life of the mine is less with long level intervals and more with short intervals. When the shaft is sunk on the dip of the orebody the cost of a crosscut is eliminated.

The cost of a shaft station, ore pockets, sump, pump station, and similar excavations, used only for the life of a single level, is chargeable to the ore mined from that level. If a pump station is constructed to serve for the life of the mine, this cost becomes a capital-expense charge.

The manner in which mining costs are related to the proper spacing of levels varies to such an extent in actual practice that any generalizations are difficult to make. The reader is referred, therefore, to several examples given hereafter which illustrate the effect of various level spacings on mining costs under different conditions.

FACTORS GOVERNING LEVEL SPACING

The factors which enter into the problem of level spacing may be classified into two major groups: (1) Geological, and (2) economical. The factors are to some extent interrelated so that the above classification is more or less arbitrary. The geological factors and their relations to proper level spacing are as follows:

Geological Factors

1. Amount of Probable Ore Developed Per Vertical Foot.- The amount of probable ore developed per vertical foot is the "common denominator" used in making cost estimates for various level spacings. The extension of the ore in depth in a producing mine can be approximated sufficiently closely for the problem of level spacing from the following data: (1) Previous production records and estimates of ore in sight on the lower levels of the mine which have a relation to the possible continuation of the ore in depth; (2) probable continuity of the orebody in depth as evinced by orebodies in other mines in the same district; (3) geological characteristics of the orebody as shown on the plan and longitudinal maps of the mine workings, and the regularity of its dip, length, width, and grade of the ore with due regard to structural irregularities, such as faults or dikes, which may have a bearing on the extension of the ore in depth; (4) prospect workings or exploratory drilling data, below the lowest working level.

There exist certain types of ore deposits where the previous yield per vertical foot of depth can be connected to the probable extension of the ore within fairly close limits. Such is the case with the Lake Superior copper lodes, the South African gold bankets, and other vein deposits where the structural conditions governing ore deposition are fairly uniform. In some other types of ore deposits where the ore occurs in scattered bunches and where the past history of the mine has demonstrated a regularity of their recurrence the probable amount of ore developed per vertical foot can be approximated closely enough for the problem of level spacing.

In some ore deposits, notably of the limestone replacement type, where the criteria are lacking for the probable continuation of the orebodies in depth, the problem of proper level spacing can be based on a cost analysis of what would have been the most economical level spacing, using the average amount of ore developed per vertical foot of depth in past operations, and applying the level interval so determined to the future developments of the mine.

2. Relative Advantage of Various Level Intervals for Prospecting.- The use of levels for prospecting predominates during the early life of a mine, and for this work the level interval is determined largely by geological conditions. The main object is to make available the greatest amount of ore at the least expense. The relative advantage of various level intervals solely for prospecting is not within the scope of this paper, and the factors herein discussed are applicable only to mines which have reached the productive stage.

In most productive mines the relative advantage of various level intervals for prospecting is not of primary importance, as within certain limits one level may serve as well as another. In some types of ore deposits which are erratic in their mode of occurrence it may be necessary to prospect the walls of the stopes frequently as mining is carried upward. In many cases the mining method is such that intermediate drifts or crosscuts may be driven from stopes without excessive expense, so that prospecting has little influence on level spacing. Where the relative advantages of various level intervals is important for prospecting, this factor must remain a matter of opinion as determined by experience with individual mine conditions, as it can not be calculated in a cost analysis.

3. Structural Features of the Ore and Enclosing Rocks.- The structural features of the ore and enclosing rocks determine largely the choice of the mining method and thus indirectly affect the level spacing. In general, where the ore deposits have a dip of 40° or more, the firmer the wall rocks the greater the level spacing, and, conversely, where the ore and enclosing rocks are structurally weak the level interval must be decreased in order to avoid unnecessary expense entailed in keeping long ore passes and raises open even though filling with waste is carried on contemporaneously with ore extraction. In some mines where the ore is hard and abrasive the cost of keeping the ore passes in repair or relining the chutes if worn out before a stope is finished may offset any advantages resulting from greater level spacing. In deposits having flat dips which do not permit the ore to run to the cars by gravity the spacing of levels is determined to some extent by the cost of transporting ore and supplies in the stopes, especially when hand shoveling is employed. With the increasing use of mechanical scraping in flat stopes a greater level interval can be employed than would otherwise be the case if hand shoveling were used. There are other structural characteristics of ore deposits and their enclosing rocks which have a bearing on level spacing, but as these features are closely related to mining costs, their influence is brought out in a cost analysis of the problem for any particular mine.

Economical Factors

The economical factors and their relation to the proper spacing of levels are as follows:

1. Time Required to Open a Level Prior to Ore Extraction.- The time required to open a new level prior to production has little influence on level spacing in large mines, because usually the development is planned far enough in advance so that there is little necessity for spacing levels according to the time required to reach the ore. In some small mines, however, where the ore reserves are small or the development has lagged behind the ore extraction, the time required to establish a new level may be the deciding factor in level spacing. This is more of a question of keeping the development far enough in advance so that

the capacity of the mill does not crowd the underground operations. On the other hand, development work may be so far ahead of ore extraction that the amount of capital tied up in idle workings may be an appreciable figure. The most economical course lies midway between the above two extremes.

2. Life of Level During Stoping Operations.- The life of a level during the stoping period has an important bearing on level spacing as the maintenance costs of keeping the workings open increase in proportion to the time required to mine the ore. The maintenance expense may amount to a considerable percentage of the mining costs under heavy ground conditions where the excavations must be artificially supported.

In some mines where the orebodies are erratic in occurrence the life of the level can not be estimated in advance because a number of levels may be kept open for a considerable period after the bulk of the ore has been mined in order to maintain bases for prospecting. Usually, the maintenance cost for keeping levels open solely for prospecting may be considered a capital charge and as such has no bearing on level spacing. In general, the most desirable and economical course is to concentrate the mining operations on as few levels as possible in order to reduce the maintenance expense, decrease the amount of supervision necessary, and to improve overall efficiency generally.

The recovery of level pillars, stope pillars, and stringers of ore should be undertaken as soon as possible after the bulk of the ore has been mined. Leaving the recovery of such "loose ends" to the future is generally an uneconomical procedure, as subsidence (in heavy ground) is sure to occur, workings will be closed, grades destroyed, and the work of cleaning up the level will otherwise be rendered expensive. Where "loose ends" are left to accumulate for mining at a future date their extraction may require heavy preparatory expense, especially if they are situated in vitiated upcast air where work is carried on at a low efficiency. In the event that pillars of ore must be left to support the hanging wall, and where it is uneconomical to substitute artificial support, their recovery at a future date is fraught with danger, and their ultimate recovery, in most cases, is extremely doubtful.

3. Production Requirements from the Mine.- In small mines where the tonnage of ore in sight is sufficient to supply the daily production requirements for only a short period, the level spacing may be chosen in accordance with the time required to reach additional ore supply. In such a predicament the proper level spacing is determined largely by necessity and not economy. Where unusual speed is essential in mining operations it generally results in increased costs, and in consequence the development work should be planned well in advance to avoid the necessity for undue haste. However, this does not always hold true, because in large mining enterprises maximum speed may be essential in order to avoid an excessive charge on capital outlay.

Any large-tonnage method of mining necessitates an exceptionally good underground transportation system; for the critical factor in the scheme of operation is getting the ore to the shaft and to the surface. The hoisting plant must be located far enough outside the zone of mining to be safe from disturbance caused by ground movements, and this may entail long haulage distances on each level used for transportation. Therefore, the cost of each haulage level must be carefully balanced against such costs as raising and maintenance which influence the spacing of levels.

4. Development and Mining Costs Which Influence Level Spacing.- The principal cost items which may enter into the cost analysis of proper level spacing are as follows:

a. Cost of shaft sinking preparations and removal of equipment after sinking operations are completed.

b. Cost of cutting and timbering shaft stations, ore pockets, sumps (when used only for the life of the level during stoping period), powder magazines, underground repair shops, or other excavations which are used only for the life of mining operations on the level.

c. Cost of drifts and crosscuts and of sub or grizzly levels which are required to make ore available for extraction.

d. Cost of installation of level equipment such as pumps, ventilating fans, track, power cables, and air and water lines, which are used during the period of ore extraction.

e. Cost of mining level pillars, floor pillars, and stope pillars.

f. Cost of raises and ore passes required to extract the ore.

g. Cost of maintenance of drifts, crosscuts, raises, ore passes, sub or grizzly levels for the life of the level.

h. Cost of hoisting ore to the surface.

i. Cost of pumping (since the level interval may increase the lift).

j. Cost of handling ore and supplies in the stopes.

k. Cost of ventilation.

l. Cost of waste filling.

m. Cost of supervision.

n. Interest on potential capital represented by broken ore tied up in storage in the stopes as in shrinkage stoping.

In most mines the foregoing cost items "a" to "e" decrease when referred to a cost per ton basis, and items "f" to "n" increase proportionately to the level spacing. The above costs are subject to a wide variation under different mining conditions so that all of them are not necessarily present in any particular mine.

The cost of shaft sinking preparations and the removal of equipment after sinking must be repeated each time a new level is opened if the sinking is done in single lifts. In some mines where the shaft must be deepened under crowded working conditions the sinking may hamper the normal routine of operations. In such a case the cost entailed by congestion due to sinking should be included in the above cost item, if it can be approximated.

The cost of a crosscut from the shaft to the orebody may be a considerable item of expense where exceptionally long crosscuts are necessary. In some cases this cost can be appreciably reduced by placing the haulage-way connection with the shaft at an increased distance and using intermediate levels not connected with the shaft as bases for stoping operations. By so doing the costs of items "a," "b," and "d" are reduced, as well as the tramming costs, because transportation is more concentrated.

In a cost analysis of level spacing the amount of drifts, crosscuts, and other workings necessary to mine the ore can not be determined accurately in advance unless the structural conditions are regular. However, in most cases, the cost of these workings on a new level may be approximated from the amount of such workings necessary in previous operations.

Only the cost of installing track, air and water lines, power cables, and similar equipment is considered in the cost analysis of these factors with respect to level spacing, because in the majority of mines this equipment is used over and over again as old levels are abandoned and new ones opened up.

The costs of hoisting, breaking ore, waste filling, handling ore and supplies in the stopes, ventilation, and supervision, ordinarily have little influence on the level spacing as such costs remain fairly constant regardless of the distance between levels. If necessary these items can also be included in the cost analysis, if they vary considerably for different level intervals or if the basic data warrant such a degree of refinement.

The amount of ore left in storage in shrinkage stoping is one of the controlling factors in the determination of proper level spacing where this method of mining is employed, as the breaking of more ore than is necessary for immediate treatment results in the investment of a considerable sum of money. An effort should be made to hold the amount of broken ore in the stopes at a minimum by mining the orebody in small sections, and emptying each section as soon as it has been stoped through to the level above. In some cases where the values in

the ore are unevenly distributed it may be necessary to mix various grades of ore from different parts of the mine in order to maintain a uniform grade to the mill for better recovery in metallurgical treatment. If the superficial area must be large in shrinkage stoping, the level interval should be such that the amount of ore left in storage will not be excessive.

5. Influence of Level Spacing on Recovery of Ore.- The influence of level spacing on the recovery of ore is an important factor in mines having walls which tend to spall or become difficult to support if the level interval is too great. Examples are mines employing the shrinkage-stope or open-stope systems of mining where large areas of the hanging walls remain unsupported. Large blocks or slabs of waste are liable to slough from the walls and contaminate the ore, resulting in a decrease in ultimate profit. Also in timbered stopes which are subsequently filled, where the ground is in constant movement and too long a period is required to mine a section, the increasing costs of maintenance may nullify any advantage gained by an increase in level spacing.

In large deep mines employing open-stope methods, air blasts and creeps may be of paramount importance in the selection of the level interval. In mines where this condition exists and must be resisted in a great degree, the spacing of levels must remain a matter of opinion as influenced by past experience with the mine conditions.

6. Safety and Health of Workmen.- Indirectly the safety and health of the workmen are reflected in the costs of mining, because the labor unit efficiency is increased by better ventilation and more freedom from the liability to accidents. During the last decade the phenomenal growth of other industries and their labor needs has diverted the stream of labor from the mines. Aside from the humanitarian principles involved, the mining industry as a whole has given more attention to improvement in underground working conditions in order to improve the efficiency of labor and to attract the better class of employees.

EXAMPLES

To illustrate the influence of the foregoing factors on level spacing, cost estimates have been prepared on three mines under different conditions. As previously stated, the determination of the proper level interval for a particular mine constitutes a special case so that the hypothetical mines herein described will serve the purpose of illustrating the problem as well as actual ones. Costs have been approximated from previous operating experience and technical publications of the Bureau of Mines. It will be observed in the following tables that costs are distributed equally for the various level spacings so that a reasonably close estimate is sufficient where detailed costs from previous records are not available. Under the law of averages the estimated costs will tend to compensate each other, providing good judgement is exercised. It may be stated that the cost analysis for proper level spacing forms only the basis for selecting the proper level interval, as the other factors which can not be reduced to a cost basis must also be considered.

Mine 1

Conditions.- In the cost analysis for proper level spacing shown in Table 3, the ore occurs as scheelite in an altered limestone bed having an average dip of 60°. The average width of the orebody is 4 feet and the length on the lowest level is 1,200 feet. The enclosing rocks are hard and firm and the mine openings stand indefinitely with little artificial support. The grade of the ore is fairly uniform. Ore is mined by the shrinkage-stope method. Entry to the mine is made by a shaft inclined 75° in the same direction as the orebody. The bottom level is 500 feet below the collar and the distance from the shaft to the orebody at this horizon is 120 feet. The mine is well ventilated by natural means, as the stopes from

the bottom level are connected through to the surface. About 20 gallons of water per minute are pumped from the mine. The tonnage of ore broken in the stopes is estimated to be 10,000 and the policy of the management is to maintain this amount of broken ore in reserve as nearly as possible. It is possible to keep the amount of broken ore fairly constant by regulating the size of the stoping sections, so that the level interval has no relation to the amount of broken ore tied up in storage in the stopes. The annual production is 40,000 tons. There is sufficient ore in sight on the lowest level to maintain the production requirements for at least eight months. The possibility of the extension of the ore in depth is excellent judging from geological conditions. The following cost analysis (Table 3) is made to determine the most economical level spacing for future development.

Explanatory Notes, Table 3

Relative Cost Estimates.- (b) Ore developed per vertical foot is estimated to be 462 tons, as determined from previous production and ore in sight on the lowest level. Estimated tons of ore developed per level is equal to "a" times 462.

(c) This item can be estimated from progress data of former development operations in the mine. In this case progress in shaft sinking is assumed to be 4 feet per day and in shaft crosscutting 5 feet per day; the time required for cutting the shaft station and ore pockets is 15 days, and for drifting and opening up the orebody prior to stoping operations is 70 days. After a certain amount of development work is done, the remainder proceeds simultaneously with stoping operations. The length of shaft and shaft crosscut varies with the level interval, but the amount of additional lateral development work necessary is assumed to remain constant regardless of the level interval.

(d) This item is obtained by dividing "b" by 40,000 (annual production).

(e) Based on tons of ore produced per year.

(f) Calculated from dip of vein (60°) and inclination of shaft (75°). Distance from shaft to orebody on lowest level, 120 feet.

(g) Calculated from dip of shaft (75°) and length of shaft required for different level intervals.

(h) The number of manways required in orebody remains constant, regardless of the level interval. Manways are spaced at intervals of 80 feet. Length of the orebody is 1,200 feet, as shown on the lowest existing level.

(i) Calculated from dip of orebody (60°).

(j) It is assumed that the chute spacing is determined largely by the desirability of keeping the broken ore at a uniform distance below the back of the stope, thus affording ready access to the back in drilling or barring down. In order to draw off the excess ore so that the top of the broken ore is kept relatively level, the chutes can not be placed too far apart, or "funneling" of the broken ore is liable to occur. Where funneling of the broken ore does occur due to improper chute spacing, the erection of staging for drilling will be required. In some cases it may be necessary to decrease the distance between chutes as the level interval is increased to minimize the tendency of the broken ore to funnel. In the problem under discussion the chutes are spaced at intervals of 20 feet for all level intervals. Three chutes are spaced equally between two consecutive manways.

(k) In the problem it is estimated that 10,000 tons of ore must remain in storage to provide working floors for stoping operations.

Estimated Unit Costs.- (l) These costs remain fixed, regardless of the level interval. Costs can be taken from previous operations and assumed in the problem to be as follows: Shaft sinking preparation and removal of equipment after sinking, $300; cost of shaft station, $900; cost of ore pockets, $500; cost of sump and installation of pump, $200; cost of powder magazine, $100.

TABLE 3.- MINE 1

Relative Cost Estimates for Establishing a Level at Various Intervals

Annual Production: 40,000 Tons. Mining Method: Shrinkage Stoping.

a. Vertical level interval....feet	60	80	100	120	140	160	180	200
b. Estimated ore developed per level .... tons	27,720	36,960	46,200	55,440	64,680	73,920	83,160	92,400
c. Time required to open level prior to stoping....months	4 1/2	5	5	5	5 1/2	5 1/2	6	6
d. Life of level during stoping operations.... do.	8 1/2	11	14	17	19 1/2	22	25	28
e. Number of levels to remain open in order to maintain production....	2	2	2	2	2	2	2	2
f. Length of crosscut from shaft to orebody for different level intervals....feet	139	145	151	157	164	170	176	182
g. Length of shaft for different level intervals....do.	62	83	104	124	145	166	186	207
h. Number of manways required in orebody....	15	15	15	15	15	15	15	15
i. Length of manways for different level intervals....feet	69	92	116	139	162	185	208	231
j. Number of chutes required in orebody....	45	45	45	45	45	45	45	45
k. Amount of ore in storage during one year....tons	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000

ESTIMATED COSTS PER TON OF ORE FOR DIFFERENT LEVEL INTERVALS

a. Vertical level interval....feet	60	80	100	120	140	160	180	200
1. Shaft sinking preparations and removal of equipment after sinking; shaft station; ore pockets; sump; and installation of pumping equipment and powder magazine....	$0.07	$0.05	$0.04	$0.04	$0.03	$0.03	$0.02	$0.02
2. Crosscut from shaft to orebody (includes cost of installation of air and water lines, track, and waste disposal)....	.05	.04	.03	.03	.03	.02	.02	.02
3. Drifts and crosscuts in orebody (includes cost of installation of air and water lines, track, and waste disposal)....	.69	.52	.42	.35	.30	.26	.23	.21
4. Chutes in orebody (includes cost of timber required; timber not recovered); belling out chutes preparatory to stoping also included....	.05	.04	.03	.02	.02	.02	.02	.01
5. Manway raises in orebody (includes cost of air and water lines, ventilation of raises while driving, timbering)....	.05	.05	.05	.05	.05	.05	.05	.05
6. Maintenance of raises during stoping operations....	.03	.03	.04	.04	.04	.05	.05	.05
7. Recovering pillars of ore between chutes....	.49	.37	.29	.24	.21	.18	.16	.15
8. Annual interest on potential capital represented by broken ore in storage in stopes....	.10	.10	.10	.10	.10	.10	.10	.10
9. Handling supplies, tramming, pumping, hoisting, supervision, breaking ore, breaking bowlders in stopes....	1.60	1.60	1.60	1.60	1.60	1.60	1.60	1.60
10. Other direct costs of mining such as shaft sinking, installation of pipe lines and power cables in shaft, drill steel sharpening, compressed air, haulage equipment, pumping equipment, ventilation equipment not included in above....	.70	.70	.70	.70	.70	.70	.70	.70
Total estimated costs of mining from various levels....	3.83	3.50	3.30	3.17	3.08	3.01	2.95	2.91

(2) Cost of shaft crosscut is taken as $10 per foot. This can be estimated from previous cost records.

(3) Cost of drifts and crosscuts is assumed to remain constant, regardless of the level interval, as the same amount of work is required per level. On the lowest level 2,400 feet of lateral development work was necessary and the average cost was taken as $8 per linear foot.

(4) Cost of chutes and belling-out over the chutes preparatory to stoping is estimated from cost records to be $30 per chute.

(5) Cost of manways in orebody is taken as $16 per foot of completed manway for the 100-foot spacing of levels. It is estimated that the cost of driving a manway will increase $1 per foot for each additional 20 feet over the 100-foot spacing and decrease $1 per foot for each 20 feet below the 100-foot spacing. In the problem the increase in the average cost per foot of manway as the level interval is increased is not reflected in the table because the amount of ore made available increases proportionately to the cost of manways. The cost for each manway for the different level intervals is as follows:

Level interval, feet	Cost per manway
60	$ 840
80	1,140
100	1,460
120	1,800
140	2,160
160	2,540
180	2,940
200	3,360

(6) Cost of maintenance of manways, due mainly to blasting and to a lesser extent to the weight of broken ore in the stopes, increases in proportion to the length of the raise. Cost of raise maintenance for the life of each raise is estimated to be as follows:

Level interval, feet	Cost per raise
60	$50
80	85
100	120
120	155
140	190
160	225
180	260
200	295

(7) Recovery of level pillars and pillars between chutes is estimated to cost $5 per ton for extracting 2,700 tons of ore, or a total of $13,500. The amount of ore recovered in pillars is assumed to remain constant, regardless of level interval.

(8) The annual interest on the capital represented by the broken ore held in storage is assumed to remain constant. The amount of ore (10,000 tons) thus held in storage represents an investment of 10 cents per ton, the average value of the ore being $8 per ton and interest on capital being estimated at 5 per cent. In this case it is assumed that the ore is mined in sections, as the amount of broken ore in the stopes remains practically constant for any level interval.

Mine 2

Conditions.- In Mine 2 the ore is gold in a quartz gangue. The vein material is fractured and structurally weak; the enclosing rocks are soft and the ground invariably heavy so that all the mine openings in proximity to the vein require timber support and constant repair to keep in good condition. Ore is mined by the square set and fill method; waste for filling is obtained from the walls. The vein has an average dip of 70° and the inclination of the shaft coincides with the vein dip on the lower levels. The shaft is 3,000 feet long and on the lower levels is 380 feet distant from the orebody. The air conditions are poor under natural ventilation so that artificial ventilation must be employed. The mine is dry on the lower levels and the only water pumped from the mine is a small amount of surface water which is caught on the upper levels.

The vein is irregular in width and length as it alternately pinches and swells both on the strike and dip. From a number of the lower levels of the mine the average amount of ore developed per vertical foot of depth was 1,150 tons. The amount of ore available is sufficient to supply the production requirements for one year. The yearly production is 90,000 tons. The following cost analysis (Table 4) is prepared to determine the most economical spacing of a new level.

Explanatory Notes, Table 4

Relative Cost Estimates.- (b) Tons of ore developed per vertical foot is 1,150. In the problem this figure is assumed to be the average yield from the three lowest levels in the mine.

(c) The number of months required to open a new level prior to stoping is based on previous progress reports; 2 months are required for constructing shaft station and ore pockets, 3 months for drifting in orebody prior to stoping, 2½ months for driving shaft crosscut, and for sinking the shaft the time required varies according to the level interval. Shaft sinking is done at the rate of 5 linear feet of completed shaft per day.

(d) Based on tons of ore available per level and annual production, or item "b" divided by 90,000.

(e) Based on items "b", "c", and "d", and yearly production.

(f) The shaft is assumed to be parallel with the dip of the orebody and 380 feet distant therefrom.

(g) Estimated from previous mining operations. The number of manways is assumed to remain constant for any level spacing.

(h) Estimated from previous mining operations. In this case the spacing of chutes is determined by economy in transferring the broken ore to the chutes by hand shoveling. Therefore, the number of chutes required to mine the ore remains constant for the various level spacings.

(i) Calculated from dip of vein (70°) and inclination of shaft (70°) for various level intervals.

Estimated Unit Costs.- (1) From previous cost records of the mine these costs are estimated to be as follows:

Cost of shaft sinking preparations and removal of equipment after sinking..................	$1,000
Cost of cutting shaft station and timbering	$2,500
Cost of ore pockets...	$1,000
Total...	$4,500

Table 4.- Mine 2

Annual Production: 90,000 Tons. Mining Method: Square Set and Fill.

RELATIVE COST ESTIMATES FOR ESTABLISHING A LEVEL AT VARIOUS INTERVALS

a. Vertical level interval..........feet	60	80	100	120	140	160	180	200
b. Estimated ore developed per level..........tons	69,000	92,000	115,000	138,000	161,000	184,000	207,000	230,000
c. Time required to open level prior to stoping..........months	8	8	8	8 1/2	8 1/2	8 1/2	9	9
d. Life of level during stoping operations.......... do.	9	12 1/2	15 1/2	18 1/2	21 1/2	24 1/2	27 1/2	30 1/2
e. Number of levels to remain open in order to maintain production..........	3	3	2	2	2	2	2	2
f. Length of crosscut from shaft to orebody for different level intervals..........feet	380	380	380	380	380	380	380	380
g. Number of manways required in orebody..........	15	15	15	15	15	15	15	15
h. Number of chutes required in orebody..........	50	50	50	50	50	50	50	50
i. Length of chutes, manways, and shaft for different level intervals..........feet	64	85	106	128	149	170	192	213

ESTIMATED COSTS PER TON OF ORE FOR DIFFERENT LEVEL INTERVALS

a. Vertical level interval..........feet	60	80	100	120	140	160	180	200
1. Shaft sinking preparations and removal of equipment after sinking; shaft station; ore pockets..........	$0.07	$0.05	$0.04	$0.04	$0.03	$0.03	$0.02	$0.02
2. Crosscut to orebody from shaft includes cost of installation of air and water lines, track, timbering, ventilation, and waste disposal..........	.07	.05	.04	.04	.03	.03	.02	.02
3. Drifts and crosscuts in orebody (includes cost of installation of air and water lines, track, timbering, ventilation, and waste disposal)..........	.72	.54	.43	.36	.31	.27	.24	.22
4. Maintenance of drifts and crosscuts for life of stoping operations..........	.10	.11	.11	.11	.11	.11	.11	.11
5. Manways in orebody (includes cost of installation of air and water lines, timbering, ventilation)..........	.13	.14	.15	.16	.17	.18	.19	.20
6. Chutes in orebody (includes cost of timbering and lining)..........	.45	.46	.46	.46	.46	.46	.46	.46
7. Maintenance of chutes and manways during stoping operations..	.15	.21	.26	.31	.36	.41	.46	.51
8. Ventilation of stopes during stoping period..........	.04	.05	.06	.07	.09	.10	.11	.12
9. Handling supplies in stopes..........	.06	.07	.08	.09	.10	.11	.12	.13
10. Extracting level pillars..........	.50	.37	.30	.25	.21	.19	.17	.15
11. Breaking ore, handling ore in stopes, timbering, supervision waste filling, tramming, and hoisting..........	2.40	2.40	2.40	2.40	2.40	2.40	2.40	2.40
12. Shaft sinking, compressed air, mining equipment, surface labor and general underground labor directly chargeable to mining costs not included in above and which remain constant regardless of level interval..........	1.10	1.10	1.10	1.10	1.10	1.10	1.10	1.10
Total estimated direct costs of mining from various levels..	5.79	5.55	5.43	5.39	5.37	5.39	5.40	5.44

(3) Cost of drifts and crosscuts in orebody for development of the known orebody is assumed to be $10 per linear foot of completed work. Cost of lateral workings driven for prospecting is not included. The average amount of development work required on the three lower levels of the mine is taken as 5,000 feet per level. It is assumed that the same amount of work will be required, regardless of the level interval, in opening up a new level.

(4) The cost of maintenance of drifts and crosscuts for the life of stoping operations is assumed in the problem to be $0.40 per linear foot per month. From previous progress reports and cost data it is estimated that on the average about 2,000 feet of lateral workings must remain open for the life of the stoping operations in order to maintain the desired production. The shaft crosscut is taken as being in fairly good ground which requires little maintenance.

(5) Cost of manways in the orebody is estimated from cost records and experience to be $10 per linear foot vertically for the first 60-foot level interval; the cost increases $2 per linear foot for each additional 20 feet thereafter. Cost of each manway for the various level intervals is as follows:

Level interval, feet	Cost per manway
60	$600
80	840
100	1,120
120	1,440
140	1,800
160	2,200
180	2,640
200	3,120

Fifteen manways are required for each level.

(6) From previous operations it is estimated that an average of 50 chutes was necessary to mine the ore from the three lower levels of the mine. The chutes are carried up as stoping is advanced so that the cost is assumed to remain fairly constant for the various level spacings. Cost of chutes is taken as $10 per linear foot.

(7) From past experience it is estimated that an average of 14 chutes and 4 manways must remain open at any time during the stoping period in order to maintain the desired daily tonnage. The same number is required, regardless of the level interval. The average cost of maintenance of chutes and manways is $1 per linear foot of raise or manway per month.

(8) Ventilation in stopes is assumed to cost $0.06 per ton for the 100-foot spacing and to increase or decrease at a rate of $0.01 per ton for each 20 feet.

(9) Cost of handling supplies in the stopes is assumed to be $0.08 per ton for the 100-foot spacing and to increase or decrease $0.01 per ton for each 20 feet.

(10) From the three lower levels of the mine it is estimated that an average of 11,500 tons of ore was mined from level pillars from each level at a cost of $3 per ton. The $3 per ton includes only the cost of breaking and getting the ore into the chutes and the timbering necessary. The amount of ore in the level pillars remains practically constant, regardless of the level interval.

Mine 3

Conditions.- In the table for Mine 3 the mine conditions are assumed to be as follows: The ore occurs as complex sulphides of lead-zinc-silver in a vein having an average dip of 80°, an average width of 10 feet, and an average length of 4,000 feet. Selective mining and

hand sorting are employed in conjunction with the cut-and-fill method of mining. The mine is entered by an inclined shaft sunk on the dip of the vein. The annual production of the mine has been 115,000 tons of ore. Mineralization occurs in alternate bands of waste and ore across the width of the deposit so that most of the ore can be broken clean by selective mining. In some places where selective mining is not feasible, hand sorting is employed to obtain a high-grade product of ore. The waste derived from hand sorting and selective mining is used to fill the mined-out portions of the vein. The ore is shoveled into chutes by hand; chutes are spaced on 40-foot centers. The chutes are carried up as stoping advances, and consist of two compartments, one for manway and the other for transferring the ore to the haulage level. Cribbing is used for timbering the chute-manway raises through the waste fill. An analysis of the bottom levels shows that the mine has produced an average of 950 tons of ore per vertical foot of depth, and from the geological data it is reasonable to assume that the same ratio of output will be maintained for the next 200 feet below the lowest existing level. The structural conditions of the walls are fairly good so that very little timber is required for support. Horses of waste occur in the vein and these are left in place if not in line with a raise. The mine drainage is 1,000 gallons of water per minute and this flow is continuous. No amount of pumping produces any effect on the flow of water pumped. It is not possible to seal off the water by grouting, so that the pumping equipment has to be moved from level to level as the mine is deepened. Ventilation conditions are excellent as the bottom levels are connected to the surface by a second exit and the air currents are controlled without difficulty.

The following cost analysis (Table 5) is prepared to determine the most economical spacing of a new level.

Explanatory Notes, Table 5

Relative Cost Estimates.- (b) Based on average width (10 feet) and average length (4,000 feet) of vein mined on upper levels and assumed continuity in depth.

(c) Based on previous progress reports of mine development: Shaft sinking, 3 feet per day; cutting station, ore pockets, sump and installation of pumps, 50 days; development in ore prior to stoping, 60 days.

(d) Based on estimated tons of ore developed per level (b) and annual production (115,000 tons).

(e) Based on "b," "c," "d," and annual production.

(f) Based on inclination of shaft and dip of orebody (both 80°).

(g) Based on previous production and development.

Estimated Unit Costs.- (1) Total gross cost of item 1 is estimated from previous cost records to be $4,000.

(2) Total drifts and crosscuts required on a level is estimated to be 5,000 linear feet at an average cost of $9 per foot. Total cost $45,000.

(3) Cost of chute-manways from previous work is estimated to be $10 per foot.

(4) Based on previous operations and production. An average of 20 raises remains open for the life of the stoping period. Average cost of maintenance of raises is estimated to be $1 per linear foot per month.

(5) Cost of pumping from various levels is based on cost of power as $6 per horsepower-month; amount of water pumped, 1,000 gallons per minute; head varies from 1,260 feet to 1,400 feet for different level spacing; overall efficiency of pumping plant, 80 per cent (reciprocating pumps, direct motor driven).

(6) Cost of stulling drifts prior to stoping and cost of extraction of floor pillars is estimated to be $12,000 from previous cost records.

Table 5.- Mine 3

nnual Production: 115,000 Tons. Mining Method: Horizontal Cut and Fill.

RELATIVE COST ESTIMATES FOR ESTABLISHING A LEVEL AT VARIOUS INTERVALS

a. Vertical level interval ....feet	60	80	100	120	140	160	180	200
. Estimated ore developed per level ....tons	57,000	76,000	95,000	114,000	133,000	152,000	171,000	190,000
. Time required to open level prior to stoping ....months	4 1/2	4 1/2	5	5	5 1/2	5 1/2	6	6
. Life of level during stoping operations .... do.	6	8	10	12	14	16	18	20
. Number of levels to remain open in order to maintain production	3	3	2	2	2	2	2	2
, Length of shaft, chute-manway raises ....feet	61	81	101	121	141	162	182	202
. Number of chute-manway raises required to mine ore	80	80	80	80	80	80	80	80

ESTIMATED COSTS PER TON OF ORE FOR DIFFERENT LEVEL INTERVALS

Vertical level interval ....feet	60	80	100	120	140	160	180	200
. Shaft sinking preparations; ore pockets; shaft station; sump, pump station	$0.07	$0.05	$0.04	$0.04	$0.03	$0.03	$0.02	$0.02
Drifts and crosscuts in orebody (includes cost of waste disposal, air and water lines timbering, and track)	.79	.59	.47	.39	.34	.30	.26	.24
. Chute-manway raises (complete)	.86	.85	.85	.85	.85	.85	.85	.85
Chute-manway raises maintenance	.14	.18	.22	.26	.30	.34	.38	.42
. Pumping	.36	.33	.32	.31	.31	.30	.30	.30
. Stulling back of drifts for stoping operations; extraction of floor pillars	.21	.16	.13	.11	.09	.08	.07	.06
. Breaking ore, handling ore and supplies in stopes, sorting, tramming, hoisting, and supervision	1.26	1.26	1.26	1.26	1.26	1.26	1.26	1.26
. Shaft sinking, compressed air, mining equipment, surface labor and general underground expenses directly chargeable to mining costs not included in above, and which remain constant, regardless of level interval	.83	.83	.83	.83	.83	.83	.83	.83
Total estimated direct costs of mining from various level intervals	4.52	4.25	4.12	4.04	4.01	3.99	3.97	3.98

CONCLUSION

The relation between mining costs and level spacing for the hypothetical mines in this paper is shown graphically in Figure 1. It will be observed that the horizontal point of tangency for the shrinkage stoping curve has not been reached with the 200-foot level spacing. The curves here shown are true only for the conditions and costs given. Each mine has an individuality of its own, and the problem of proper level spacing must be worked out for each particular case. After the cost analysis of the proper level spacing has been worked out the other factors which can not be reduced to a cost basis must also be considered. Although the problem is complex and involves a number of variables, the engineer's experience with the ore occurrence, mine conditions, and familiarity with the detailed cost records for his particular mine will enable him to establish the levels to the best advantage after cost estimates are prepared. The possible savings effected by proper level spacing warrant careful investigation. In making a cost analysis the engineer must place himself in the position of a contractor, personally responsible for a loss. The question of cost estimates then ceases to be academic and becomes practical. The cost data are more reliable in proportion to the length of the period over which they are taken. Any assumptions made on cost data relating to a short period must be used with discretion, especially when they represent a stage of advantageous operation. Costs at other mines are of much assistance if due allowance is made for a divergence in geological conditions and geographical environment.

This article has been prepared with the hope that it will lead to discussion and to more exact consideration of the problem of proper level spacing.

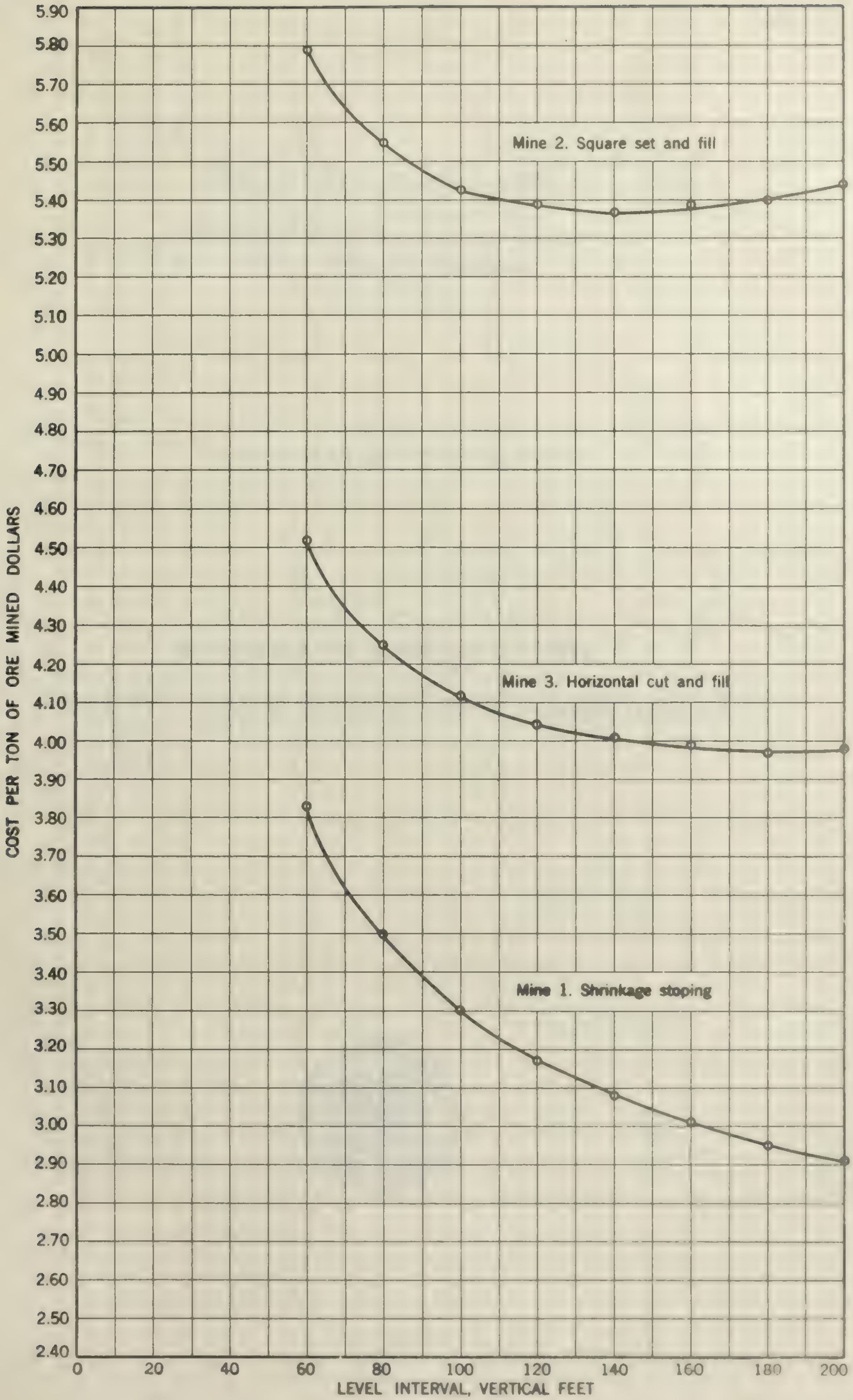

Figure 1.—Relation between mining cost and level spacing

I. C. 6614 May, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

A VENTILATION STUDY OF THE GRACETON COAL & COKE CO. MINE, GRACETON, PA.

BY

R. D. CURRIE AND E. R. MAIZE

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

A VENTILATION STUDY OF THE GRACETON COAL & COKE CO. MINE, GRACETON, PA.[1]

By R. D. Currie[2] and E. R. Maize[3]

Good ventilation is one of the first requisites in the safe and efficient operation of a coal mine. If a mine is gassy, the uninterrupted circulation of an adequate supply of fresh air is imperative if that mine is to operate with a relative degree of safety. That ventilation is not only a safety consideration, but is an extremely important economic factor is shown by the fact that in many mines 50 per cent of the total power cost is for ventilation.

The system used in the Graceton Coal & Coke Co. mine, Graceton, Pa., is believed to be unusual in many ways. Its application in this mine has resulted in good ventilation of the working faces, no interruption to the ventilating current through the use of doors, a large portion of the air circulated by the fan reaching the working places, low velocity air currents on traveling and haulage ways, low water gage and power consumption of the fan, and well-ventilated gob areas.

ACKNOWLEDGMENTS

This investigation was made possible through the courtesy of W. H. Robertson, mine superintendent. Fred Roberts, mining engineer, and Floyd Phillippi, mine foreman, assisted in the collection of information.

GENERAL INFORMATION

The Graceton mine is a slope, opening the Upper Freeport or E bed, on the Pennsylvania Railroad about midway between Blairsville and Indiana at Graceton, Indiana County, Pa. The production is approximately 1,000 tons of coal per day and about 170 men are employed underground. The mine is considered gassy and was found to be liberating approximately 300,000 cubic feet of methane in 24 hours. The return airways terminate at a circular, concrete-lined shaft 12 feet in diameter and about 115 feet deep, at which is the exhaust fan.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6614."

2 - Associate mining engineer, Pittsburgh Experiment Station, U. S. Bureau of Mines, Pittsburgh, Pa.

3 - Junior mining engineer, Pittsburgh Experiment Station, U. S. Bureau of Mines, Pittsburgh, Pa.

Mining is done by a block system with flat or level entries turned off the main slope at intervals of 1,350 feet, off which the butt headings are turned up the grade parallel to the slope. Rooms, which are about 225 feet long and 12 feet wide, are turned off the butt headings on 80-foot centers, leaving a block about 68 feet thick between rooms.

Haulage on the main slope is by rope, with the hoist on the surface. This hoist is electrically driven through a 500-h.p., 2,300-volt motor. The rope used in this haulage is 1-1/8 inches in diameter. There are five electric trolley locomotives - one 10-ton and four 6-ton - which do all of the hauling in the flat and butt entries.

Before it is blasted all of the solid coal and as much of the pillar coal as can be cut with safety is undercut by permissible mining machines. Shooting is done by shot firers who use permissible explosives, electric detonators, and dry-cell, nonpermissible batteries.

In rooms where conditions are normal, timbering consists of two rows of posts on 4-foot centers. Posts are required to be within 6 feet of the face in machine-cut places and within 3 feet of the face in hand-mined places. Timbering on entries, where required, is of two types: a bar hitched into the ribs and supported by the coal, or 3-piece sets. The legs of the 3-piece sets used on the slope are set well back into the rib to prevent their being knocked out by a wreck.

Power is carried into the mine through a borehole at 2,300 volts alternating current and is converted into 250 volts direct current in a well-ventilated substation. The main pumping station is operated on 2,300 volts alternating current. There are two 3-stage centrifugal pumps in this fireproof, well-ventilated station.

A separate manway is provided from the slope mouth to the bottom of the main slope, with overcasts at points where it crosses the haulage road. In addition to this, man trips are run at the beginning and end of the shift. When the man trip is being made up, a safety rope is encircled about the cars making up the trip and is attached to the main hoist rope to prevent any of the cars from getting away from the trip in case a coupling breaks or comes loose.

Permissible electric cap lamps are used by all men underground for illumination, and no open lights, matches, or smoking materials are permitted in the mine. Permissible flame safety lamps are carried by officials for inspecting and testing.

The mine has not been rock-dusted, nor is water used to allay the dust made in the mining operations.

THE FAN

The ventilation is induced by a Jeffrey 4½ by 7 foot double-inlet centrifugal fan housed in a brick building with concrete floors, at the

Figure 1.—Ventilation map of Graceton mine.

side of a circular shaft. A steel casing and air duct, provided with explosion doors and reversing doors, connects the fan to the shaft. The volume of air measured several times at the foot of the air shaft was 67,125 cubic feet per minute, without appreciable variation. The fan is driven by a variable-speed induction motor, operating at 2,300 volts alternating current. This motor is rated at 150 or 50 hp., and at the time of this investigation it was driving the fan through a Morse silent drive chain at 88 r.p.m., against a water gage of 0.7 inch. The fan, which is equipped with a thermostat relay on each bearing, has a system of signal lights attached to the cut-out to give warning in case it stops.

The fan at this mine, however, is not equipped with an emergency drive, nor is it supplied with a duplicate motor.

For greater safety in the operation of a mine fan, and to assure the least possible delay in getting the fan in operation again should the main power source fail, or in case of damage to the main driving motor, it is advisable to provide an emergency drive at the fan. At some mines this consists of a duplicate motor, either set up so that it can be coupled to the fan drive through a belt or clutch, or merely set in the fan house where it can be quickly placed on the foundation of the regular driving motor; at other mines the emergency drive consists of a high-power gasoline or oil-burning engine; at others it consists of a gasoline-engine driven generator with sufficient capacity to drive the fan temporarily and supply power for the man hoist.

In any event, more than one source of power should be available for driving a fan at any mine known to give off explosive gas.

SYSTEM OF VENTILATION

The ventilating system is as nearly "foolproof" as it is possible to make a complicated problem, such as the ventilation of a gassy coal mine naturally must be.

The system has primary and secondary splits, with a split of fresh air for each active room entry. There are no ventilating doors on any haulage road, thus practically insuring a continuous supply of fresh air to all active workings at all times; at any rate, the numerous dangers due to doors on haulage roads are avoided.

The main slope, with its four parallels which are intake airways has a total cross-sectional area of approximately 360 square feet over most of its length. (See Figure 1.)

The only place in the entire system where high-velocity air is traveling is in the upcast shaft. This shaft, which is 115 feet deep and 12 feet in diameter, is circular and is lined with smooth concrete with no projections. The circular shape of this shaft presents the least possible rubbing surface for the cross-sectional area obtained, which, in addition to the smooth

concrete lining, offers the least possible resistance at this one point of high-velocity air.

The first primary split is above J flat, the second at N flat, the third at P flat, and the fourth at R flat, while the fifth goes to the main dip. The first split ventilates the section O flat to the north of the main slope, where work is done on the night shift only. The air from this split rejoins the main intake at P flat. The second or N flat split is divided into three secondary splits, each ventilating a producing butt entry with returns direct to the fan over the gobs at the top of the butt. The third or P flat split ventilates inactive workings in P flat. The fourth or R flat split is divided into six secondary splits, each ventilating a separate butt heading; the fifth or main dip split ventilates only the workings in that heading, and returns through airways and overcasts to the back end of R and P flat, thence goes through numerous airways of large area to the fan.

The total number of splits in the ventilation system, including primary and secondary splits, is 14, of which 5 are primary splits and 9 are secondary splits.

All haulage roads and traveling ways are on intake airways, and no power, telephone, or signal lines are on return airways.

The longest distance traveled by any of the air--in the main dip split--is slightly over 4 miles; the total distance traveled by the air in the shortest split is somewhat less than $2\frac{1}{2}$ miles.

The calculated power, based on the formula, Work = quantity x pressure, shows that only 7.41 hp. is required to ventilate this mine on the present basis. This low power consumption is accounted for in the large cross-sectional area of airways, low-velocity air currents, and highly efficient use of the air being circulated. If this fan were operating against a water gage of 3 inches, as fans are at many mines, the power consumption would be 31.56 hp. for the same quantity of air. Then again, if the quantity of air had to be increased to twice the present amount, then the velocity would be doubled, and the pressure would be four times as great, which would make a tremendous difference in the power consumption as well as in the ventilating cost.

EFFICIENT USE OF AIR IN CIRCULATION

Measurements taken at the intake and return of the active workings show that over 80 per cent of the total quantity of air circulated by the fan is used effectively to ventilate the active working places. The air which ventilates the rooms moves directly over the pillar line in that heading and then goes into the return airways.

The flat headings are all driven to the property barrier pillar, and the last butt headings, together with the rooms driven off them, are left standing without the pillars being pulled. This insures a large open airway

direct to the fan for all air passing over or through caved regions. Pillars are extracted on a retreat system, so that no air which has passed over a caved region ventilates any active workings.

The following quotation taken from "Safety in Coal Mining"[4] is used as a basis for comparison:

> ****Tests have shown that in 16 Illinois coal mines, only 7.6 to 33 per cent (or an average of 18.6 per cent) of the air leaving the fans reached the last crosscuts nearest their respective faces, because of leakage at doors and stoppings, and through crevices in pillars and rooms. Good practice requires that at least 50 per cent of the air entering the mine shall reach the faces.

Table 1 of air readings taken in this mine shows that over 80 per cent of the air being circulated by the fan is being used to ventilate the working faces:

Table 1. - Air distribution in the Graceton mine

Place	Men	Cubic feet	
		Intake	Return
Main intake and return opposite J flat . .	145	67,266	67,125
Main dip split at last crosscut	8	6,470	7,650
3 dip at last crosscut	3	5,500	6,500
R flat split between 4 and 5 butts	44	24,800	21,300
2 butt N flat at first room	14	4,050	(See note)
3 butt N flat at first room	14	8,450	(See note)
4 butt N flat at first room	8	4,920	(See note)
Total quantity of air ventilating active working places	-	54,190	

Note: The return of N flat butt headings over gob areas made it impossible to make accurate readings on these returns.

The quality of air provided for each man is well in excess of the requirement of the Pennsylvania mining law, which is 200 cubic feet of air per minute for each man working on a split. There are 145 men underground on day shift and 54,190 cubic feet of air is supplied to the working faces, an average of 372 cubic feet per minute per man.

The high efficiency attained in the ventilation of this mine is due to:

1. The utilizing of large-area airways.
2. Coursing the air in natural lines from the slope mouth to the fan.

4 - Rice, G. S., Safety in Coal Mining; Bull. 277, Bureau of Mines, 1928, p. 27.

3. Splitting the air as soon as possible after it enters the mine.
4. Eliminating doors in the ventilating system.
5. Providing smooth-lined airways of the least possible rubbing surface wherever high-velocity air currents are necessary, such as in the upcast shaft.
6. Using substantial, fireproof stoppings and overcasts.
7. Properly regulating the air currents in the various splits.

DOORS

Doors are not used in the ventilating system of the Graceton mine.

The promiscuous use of doors in a ventilating system, regardless of efficiency of installation, always results in air loss through leakage. They also present the hazard of being blocked open--many of them are equipped with "snibs" or latches for this purpose--thus allowing accumulations of gas at working places; and they also cause increased hazards to haulage, such as wrecked doors, men being pinned by door and trip, or men encountering trips or cars when passing through doors.

OVERCASTS AND STOPPINGS

Overcasts are constructed of tile and reinforced concrete. The tile is used for the walls; the floor of the overcasts is of poured concrete reinforced with used rails. This type of construction makes a substantial overcast, and when properly hitched into floors, ribs, and roof, offers little opportunity for air leakage.

The approaches to the overcasts are smooth rather than abrupt, as is general in mines. In most cases considerable money can be saved in the initial cost of construction, in addition to a saving in power in ensuing years, if a large portion of the material shot from the top in making the overcast is used in building approaches to the wing walls.

Stoppings in this mine which are built of tile have been found to give satisfactory results because there is little roof movement and no appreciable heaving of the bottom. In mines where either of these conditions are present it would be more advisable to use brick, concrete block, or poured concrete for stopping construction.

GAS INSPECTION

A preshift inspection of the mine is made by a fire boss who uses a permissible flame safety lamp. This examination starts at 3 a.m. and the condition of the mine is reported to the mine foreman before the man trip is permitted to start into the mine. A second examination is made by the fire boss after the men enter their working places. Written reports of these inspections are made in a special book provided by the State Department of Mines.

Permissible flame safety lamps are also carried by the mine foreman, assistant foremen, shot firers, and machine operators for testing and inspecting the working places.

It is estimated that each working place is examined four times per shift by men carrying permissible-type flame safety lamps.

AIR SAMPLES AND ANALYSES

Air samples were collected by the writers and were analyzed in the gas laboratory of the Bureau of Mines in Pittsburgh, Pa. Table 2 gives these analyses:

Table 2. - Air analyses from the Graceton mine

Bottle	Place	Per cent				Air per minute, cubic feet	CH_4 per 24 hours, cubic feet
		CO_2	O_2	CH_4	N_2		
403	Return 3 dip at R flat	0.14	20.58	0.43	78.35	7,650	47,369
404	Return main dip at R flat	.15	20.56	.80	78.49	7,857	90,513
418	Top 5 butt at regulator	.12	20.53	.18	79.17	6,500	16,848
419	Top 7 butt, return 7 and 8	.15	20.46	.32	78.95	5,950	27,418
420	Top 9 butt at regulator	.12	20.46	.59	78.83	6,880	58,452
423	Full return at fan shaft	.14	20.62	.33	78.91	67,125	318,978
858	do	.11	20.58	.30	79.01	67,125	289,980

DISCUSSION OF AIR ANALYSES

All of the samples show normal oxygen and carbon dioxide for return mine air samples. Bottles 404 and 420, taken from the returns of the main dip and 9 butt R flat, indicate that the methane content is slightly higher in these two returns than good practice allows. It is advisable, wherever possible, to keep the percentage of methane in the return airways below 0.50 per cent. To accomplish this, an approved type of methane detector, sensitive enough to indicate percentages well below 1 per cent, should be used to test the returns at least once a month, preferably each week, and prompt action should be taken to increase intake air flow where methane in the return is found to exceed a safe amount.

The total quantity of methane being liberated in 24 hours is not excessive when we consider that many mines liberate twice this amount, and some mines are liberating 10 times this quantity of methane every day. Three hundred thousand cubic feet of methane, however, when properly mixed with air, would fill almost 9.5 miles of entry with explosive gas at its maximum explosive point; moreover, practically all of the most disastrous mine explosions have occurred in mines which give off less than 500,000 cubic feet of methane per day.

During this investigation a check was made at the fan to determine the variation in the percentage of methane being liberated by the mine at various times during the day. For this purpose a Union Carbide Co. permissible methane detector was used. The check was started at midnight following a day during which the mine had been idle, and was continued through the following day during which the mine worked, up to 10 p.m., when the cutters, who work on the night shift, had completed their work. Readings were taken at half-hour intervals during this period, and the instrument was reset in fresh air outside the fan housing just prior to taking each reading.

Table 3 shows the variations in methane during the period of this test. The fan ran at a constant speed of 88 r.p.m. and the water gage did not vary above or below 0.70 inch during the entire test.

Table 3. - Methane detector readings in fan housing at top of upcast shaft, December 18, 1931

Time	Methane, per cent	Activities in mine
a.m.		
12:01	0.30	Night pumper only.
12:30	.30	do.
1:00	.25	do.
1:30	.32	do.
2:00	.28	do.
2:30	.30	do.
3:00	.30	Fire-boss inspection.
3:30	.30	do.
4:00	.30	do.
4:30	.30	do.
5:00	.30	do.
5:30	.25	do.
6:00	.30	do.
6:30	.30	Man trip enters mine.
7:00	.30	Day shift starts.
7:30	.30	Day shift at work.
8:00	.35	do.
8:30	.37	do.
9:00	.32	do.
9:30	.30	do.
10:00	.30	do.
10:30	.25	do.
11:00	.28	do.
11:30	.30	do.
12:00	.30	Noon hour.

Table 3. - Methane detector readings in fan housing at top of upcast shaft, December 18, 1931 (Continued)

Time	Methane, per cent	Activities in mine
p.m.		
12:30	0.30	Day shift at work.
1:00	.30	do.
1:30	.25	do.
2:00	.25	do.
2:30	.31	do.
3:00	.28	do.
3:30	.31	do.
4:00	.25	Day shift out - cutting started; night shift working.
4:30	.28	Cutting and night shift pumpers.
5:00	.30	do.
5:30	.25	do.
6:00	.28	do.
6:30	.30	do.
7:00	.25	do.
7:30	.20	do.
8:00	.22	do.
8:30	.22	do.
9:00	.28	do.
9:30	.28	do.
10:00	.27	do Cutting finished. Night shift pumpers only.

Figure 2 shows the variation in the amount of gas liberated by the mine during the period of this test in 1,000 cubic feet of methane per hour at each half-hour period of the test.

The calculations used to compile the chart were based on 67,000 cubic feet of air being circulated by the fan throughout the entire period. That this assumption is correct is based on the facts that the temperature throughout the period of the test did not vary more than a few degrees, and that the difference in elevation between the top of the fan shaft and the slope mouth is slight, so that there would be essentially no change in the tendency for natural ventilation to counteract or assist the fan differently at various times during the test. The fan ran at a constant speed, and the water gage did not appear to vary during the entire period.

The fact that there are no doors anywhere in the mine makes it improbable that there would be any wide variation in the ventilation of the various parts of the mine during the period of the test, so that any variation in the percentage of methane traveling in the air current is likely to be due wholly to mining operations or to natural conditions.

The high peak between 7:30 and 9:30 a.m. is undoubtedly caused by the greater activity of all the miners and shot firers at this period of the working shift. All of the blasting is done during the day--most of it in the first few hours after the shift starts work--and blasting is unquestionably one of the greatest factors in the liberation of methane in the entire mining operation.

CONCLUSION

While it is realized that the ventilating system in use at Graceton mine is not perfect and that improvements can be made, there is little doubt that it is far superior to ventilating systems in common use in coal mines. It is believed that any coal mine can handle its ventilating problem as efficiently if sufficient forethought and planning are used in laying out the mine, taking into consideration the essential requirements for economic and safe ventilation.

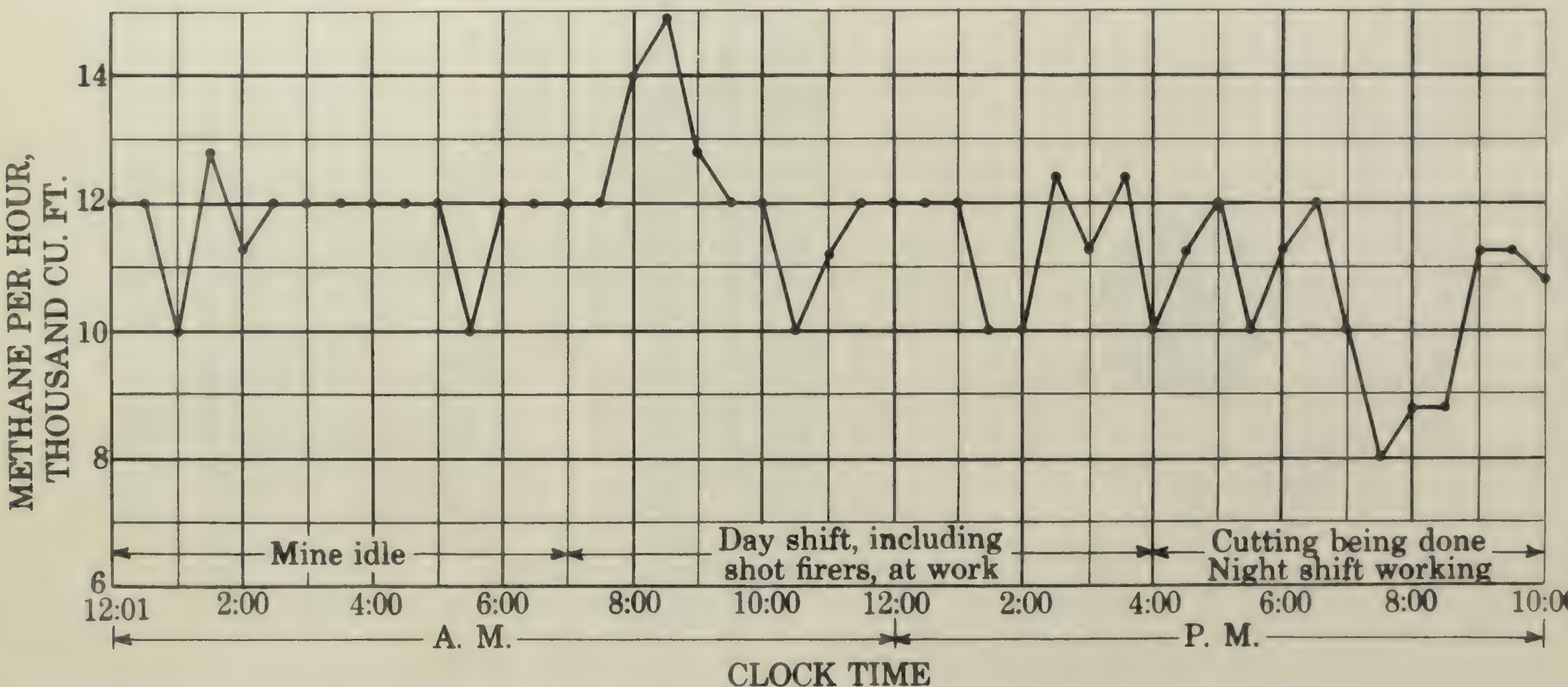

Figure 2.—Cubic feet of methane liberated per hour at each half-hour period

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UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MICA-MINING METHODS, COSTS, AND RECOVERIES AT NO. 10 AND NO. 21 MINES OF THE SPRUCE PINE MICA CO., SPRUCE PINE, N. C.

BY

H. M. URBAN

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MICA-MINING METHODS, COSTS AND RECOVERIES AT NO. 10 AND NO. 21 MINES OF THE SPRUCE PINE MICA CO., SPRUCE PINE, N. C.[1]

By H. M. Urban[2]

INTRODUCTION

This paper, one of a series being prepared for and published by the United States Bureau of Mines on methods and costs of mining at various mica mines in the United States, describes the methods now in use in the principal mica mines of the Southern Appalachian States.

These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total costs. It is recognized that publication of total costs might in some instances cause embarrassment to individual producers as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

Inasmuch as this is the first circular concerning mica mining, some information is included with reference to the occurrence of block mica and the history of mica mining in North Carolina, together with a few notes on the physical properties and uses of the mineral, and methods of processing it for use in various industries. The two mining operations described are similar, but the difference in the character and value of the products from the two mines affords instructive comparisons.

ACKNOWLEDGMENT

Acknowledgment is made to F. W. Horton, mining engineer, U. S. Bureau of Mines, for assistance in the preparation of this paper.

PHYSICAL PROPERTIES OF MICA

Mica crystallizes in the monoclinic system, commonly showing rhombic or hexagonal outlines. Because of its eminent basal cleavage it is easily split into thin sheets or films. Among its physical properties it possesses transparency, toughness, flexibility, resonance, resistance to heat, and a remarkably high dielectric strength. The chemical formula of muscovite is given as $H_2KAL_3(SiO_7)_3$. It has a hardness of 2 to 2.5 and specific gravity of 2.76 to 3.0.

MICA-BEARING AREAS IN NORTH CAROLINA

The mica producing areas of North Carolina may be grouped in three major fields. Two of these lie close to the Blue Ridge Range, while the third is to the southeast of the moun-

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6616."

2 - One of the consulting engineers, U. S. Bureau of Mines, and vice-president and general manager, Spruce Pine Mining Co.

tains in the foothills and Piedmont area. In the southwestern part of the State, the mica mines in Macon, Jackson, and Haywood Counties constitute one field. Eighty miles to the northeast, paralleling the trend of the mountains, there is another mica producing area in the counties of Yancey, Mitchell, and Avery. This field really extends northeastward, into Watauga and Ashe Counties, where numerous dikes of economic value occur. The Piedmont field is confined principally to parts of Rutherford, Cleveland, and Lincoln Counties. Regarding each of the three fields designated, it should be said that there are many commercial occurrences of mica outside of the counties named above. In fact, 20 counties in North Carolina reported a production of mica during the past three years.

The importance of North Carolina as a mica producing State is evidenced by the fact that for the 2-year period 1929 and 1930 it contributed 57 per cent of the total production of sheet mica in the United States. Of this 57 per cent, the area centering around Spruce Pine (see fig. 1) contributed more than 85 per cent, or approximately 50 per cent of the entire domestic production.

EARLY MICA MINING IN NORTH CAROLINA

Mica mining in North Carolina was carried on by some primitive race before the advent of the white man. Evidences of work done by these aborigines still remain.

While there has been a constant production of mica in the United States since about 1800, North Carolina appears to have entered the mica mining field a year or two after the Civil War, when economic changes reduced the market for agricultural products and the mountain people sought other sources of income. Mica, because of its physical characteristics, is a mineral that attracts attention, and the mountaineers soon found that it could be mined and prepared for market by simple methods. On account of its high value per pound, it could be transported for long distances, even over the poor roads existing at that time, and still return a profit. Thus the mica industry of North Carolina had its beginning. From 1911 to 1930, inclusive, the State produced more than 13,400,000 pounds of sheet and punch mica valued at about $4,400,000 and approximately a million dollars worth of scrap mica.

SOME USES OF SHEET MICA

The earlier demand for mica was as thin transparent sheets for glazing stove fronts. In this form it has sometimes been erroneously called "isinglass," which is a preparation of nearly pure gelatin.

Mica became an essential material in industry with the development and use of electricity. Its perfect cleavage, transparency, flexibility, dielectric strength, resistance to high temperatures and certain other characteristics, make it invaluable for many uses. No synthetic or natural material has yet been found to replace it in numerous applications. Its utilization in the telephone, phonograph, electric condensor, electric generating and motive equipment, and electric heating elements has been invaluable.

The demand for mica has become securely established and should increase in proportion to progress in the electrical industries. The mica industry has not, however, done much in the way of research to find new forms and applications for its products, although it appears that further uses for mica might be discovered and developed.

TOPOGRAPHIC FEATURES OF THE SPRUCE PINE DISTRICT

The Spruce Pine district is part of the elevated basin lying between the Blue Ridge on the east and the Unaka Range on the west and is drained by the Toe River. Figure 1 is a

Figure 1 - Principal mica mines in the Spruce Pine district

sketch map which shows the principal topographic features of the district and the mines which have been the leading producers of mica.

The Toe River leaves the district at an elevation of about 2,300 feet. Much of the terrain lies between that elevation and 3,300 feet. In this area many peaks rise from 4,000 to 5,000 feet high, dominated by Roan Mountain in the Unaka Range with an altitude of 6,313 feet, Celo at the north end of the Black Mountain spur 6,351 feet high, and Mount Mitchell 12 miles south rising to 6,684 feet.

The rivers have cut meandering courses, and have left surface features that are prominent but seldom precipitous. Residuals from weathering and erosion cover most of the bedrock, sometimes to considerable depth, and round the outlines of the smaller ridges and hills The whole area has been generously wooded, deciduous and evergreen trees and shrubs comprising the forest growths. These conditions make prospecting difficult and expensive. Most of the pegmatites thus far discovered have surface exposures or have been found near the surface by accident. here are probably at varying depths many other pegmatites of great value which remain to be discovered.

The Clinchfield Railroad enters the district from the northwest through the valley of the Toe River and climbs over the Blue Ridge a short distance from Spruce Pine in its progress to the southeast and the seaboard. An excellent system of paved highways traverses the region.

GEOLOGY

The rocks of the Spruce Pine area (see fig. 1) which lies westward of the main range of the Blue Ridge and north of the Black Mountain spur, are principally metamorphic and of both igneous and sedimentary origin. It has been estimated that a minimum of 10,000 feet and perhaps more than twice that depth of material has been eroded from this region since the upheaval of the mountain ranges. The rocks which are now exposed were therefore formed or have existed at great depths below what was once the land surface.

The name Carolina gneiss has been given to the most abundant and oldest of the country rocks in this region, which include such specific formations as mica, garnet, and kyanite schists and granitic gneisses. The name Roan gneiss has been applied to the more recent metamorphic country rocks in many of which hornblende is often a prominent constituent.

Intrusive bodies of granitic material and pegmatites are interbanded with, or have cut through the older country-rock structures. The larger pegmatites are generally conformable with the strike of the major folding of the older rocks and parallel the local trend of the ridges. Certain of these larger pegmatites yield block or sheet mica (muscovite) and feldspar.

The map, Figure 1, which embraces an area of about 180 square miles, shows the location of 24 mica-bearing pegmatites in which mining operations have been actively carried on, but it can be authoritatively stated that there are several hundred pegmatites in this area which are now or at some future time will be worthy of exploration for mica or feldspar or both of these minerals.

Pegmatites as represented in this area are coarsely crystallized masses of quartz and feldspar with minor accessory minerals, and may or may not contain mica in commercial sizes and qualities. The pegmatites are extremely irregular in shape, varying from pipe-like bodies to lenticular or tabular forms, depending largely on whether they cut through the enclosing rocks or conform more or less with their folding.

It may be of interest to give here the definition of a pegmatite as recently described by Frank L. Hess of the United States Bureau of Mines and his classification of pegmatites into three main groups:

"Pegmatite: A general name for rocks with coarsely crystallized and unevenly segregated minerals in dikes, veins, or metamorphic masses. Pegmatite dikes are intrusions of very wet magma and contain only granitic minerals.[1] Pegmatite veins are deposited in fissures by the aqueous solutions expelled from a freezing magma, and may be accompanied by wide replacement of the wall rocks.[2] Metamorphic pegmatites are formed by similar solutions through the replacement or alteration of previously existing minerals. Pipes are included in this class.

As pegmatites are formed only from wet magmas they are rare among the basic rocks such as amphibolites and pyroxenites and are not formed from lavas. They are found with granite, diorite, nepheline syenite and related deep-seated rocks. All sheet muscovite comes from metamorphic pegmatites and all phlogopite from replacements along pegmatite veins.

The largest known crystals and many of the rarer minerals, elements, and gem minerals are found in the metamorphic pegmatites.

As soon as pegmatite dikes are formed, replacement and metamorphism begin as shown by the introduction of tourmaline, garnet, and other minerals. Further replacement or metamorphism depends on the length of time that the solutions flow."

Potash feldspar and quartz are original constituents in all pegmatites, and in general all accompanying minerals such as albite, beryl, tourmaline, and all large micas are secondary. Small micas may be primary or secondary. Seventy or more minerals have been found in the pegmatites of North Carolina. A few of them in addition to quartz and the various feldspars (orthoclase, albite, microcline, and oligoclase) and micas (muscovite and biotite) which may be of possible commercial importance are: Allanite, auerlite, beryl (emerald, golden beryl, aquamarine), columbite, cassiterite, corundum, crytolite, garnet (almandite, pyrope), hiddenite, kunzite, kyanite, monazite, rutile, samarskite, topaz, tantalite, various uranium minerals (uraninite, clarkite, gummite, uranophane, autunite, etc.), and zircon.

HISTORY OF THE NO. 10 MINE

While the presence of the pegmatite at the No. 10 mine has been known for many years, its probable value as a source of sheet mica first attracted attention in 1903 when a heavy freshet uncovered a portion of it, exposing blocks of mica of commercial size and quality.

During the following 10 years, several attempts were made at mining, and the pegmatite was penetrated for about 150 feet. Operation by hand methods and the use of steam drills and inferior dewatering devices did not prove profitable, and work was abandoned until 1925 when the Spruce Pine Mica Co. bought the property.

With proper equipment the mine has proved profitable and has supplied the company's factory at Spruce Pine with a wide variety of crude mica for processing. This mica has been exceptionally large. One block or crystal was taken out that weighed 4,320 pounds and was approximately 3 by 3½ feet in cross section and 32 inches thick.

The income from the property has been materially increased by the sale of considerable quantities of both potash and soda spar, the latter predominating in those portions of the pegmatite that have thus far been worked. The following analyses represent mill grades of these two spars.

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1 - A dike as ordinarily understood is a crack or fissure filled by molten rock.

2 - A vein is a tabular filling of a crack or fissure by minerals deposited from solution.

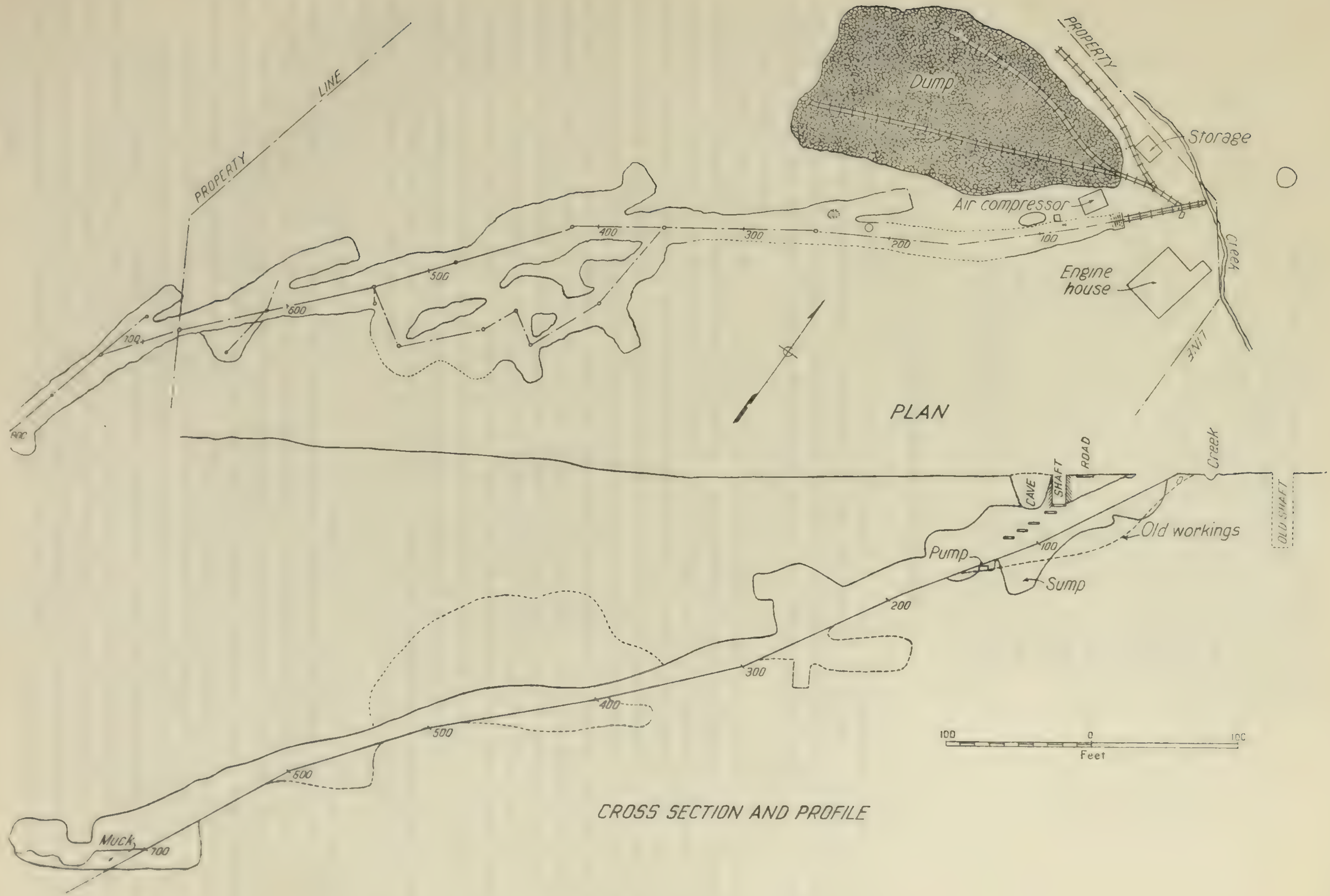

Figure 2.—Plan and cross section of Spruce Pine No. 10 mine

Analyses of feldspar from No. 10 mine of the Spruce Pine Mica Co

	Potash spar, per cent	Soda spar, per cent
SiO_2	66.2	67.84
Al_2O_3	18.53	19.73
Fe_2O_3	.07	.09
CaO	.18	.83
MgO	nil	Trace
K_2O	12.06	1.37
Na_2O	2.10	9.78
Loss on ignition	.27	.38

Description of No. 10 Mine

The No. 10 mine is about 1½ miles by road directly south of Newdale, Yancey County, N. C., on the eastern watershed of South Toe River from which it is about ½ mile distant (see fig. 1). Spruce Pine, where the company's mica processing plant is situated, is the largest town in the district, and is about 8 miles east of Newdale, from which it is reached by a concrete highway.

The pegmatite at the No. 10 mine is of the metamorphic type. It has an irregular cross section and a general trend 47° west of south. Its average pitch is about 25°. A plan and cross section of that portion of the orebody which has been mined are shown in Figure 2. The principal minerals in their order of abundance are albite, quartz, orthoclase, and muscovite. These have generally occurred throughout the mined portion of the pegmatite in such large crystals and masses that they could be readily separated by hand, although in some places, particularly along the footwall, granitic material is prominent. Quartz occurs in large masses and with considerable continuity throughout the 800 feet along the slope to which the pegmatite has been opened. The location and trend of the massive quartz bodies are carefully observed, as the mica values sometimes follow them closely and they often define the limits of mining toward the footwall. However, the theory often expressed that the richer parts of a mica-bearing pegmatite are near one or the other of the enclosing walls or closely associated with massive quartz, was not uniformly borne out in this mine.

Of the two feldspar minerals, both of which occur in commercial quantities, albite largely predominates, plainly indicating the extent to which secondary mineralization has progressed.

The content of block mica (muscovite) in the pegmatite, as indicated by recoveries, is roughly 2 per cent, but as this does not include mine scrap and much small mica that goes to the dump, the total mica content of the pegmatite is perhaps from 4 to 6 per cent.

Besides these principal minerals the pegmatite contains epidote, thulite, garnet, monazite, cyrtolite and an unusually large variety of uranium minerals including uraninite, clarkite, gummite, uranophane and autunite. The waste dump at the property has been a bonanza for mineral collectors and many rare and beautiful specimens have been obtained from it.

A ledge of massive quartz standing almost vertically appeared at the outcrop. From 4 to 6 feet of pegmatite consisting largely of commercial feldspar and carrying block mica, stood between this quartz and the hanging wall on the west. Recently exploration on the east side of the massive quartz near the surface revealed that while the body of pegmatite is large and well defined, it is of a granitic type unfavorable for the occurrence of commercial mica. The character of the pegmatite may change, however, and further investigation of this area will be made.

As mining progressed the position and trend of the massive quartz were seldom as conformable to the hanging wall as near the outcrop. In some cases, the quartz occupied the full width of the pegmatite with feldspar above and below, while at others it occurred as large or small isolated bodies.

The westerly or hanging wall usually has a steep dip to the northwest, but in the upper workings it curves toward the horizontal, and between stations 400 and 550 it flattens out over the area mined. The wall is a somewhat altered mica gneiss, showing greater schistocity along the contact, but giving little evidence of disturbance. For intervals of 40 to 50 feet it stands smooth as a masonry wall; at other places it swells into the pegmatite for several feet.

The footwall, also of mica gneiss, probably of the Roan series, is quite irregular, projections of the wall entering the pegmatite in many places. Occasionally large fragments or horses of country rock are found near the footwall, and granitic phases are also frequent along this wall.

At several places there were seams normal to the enclosing walls, which showed some water seepage and an increase of the commoner accessory minerals in their vicinity. The rare minerals were always found near the hanging wall.

It is the rule in this section at least, that the mica in pegmatites occurs in roughly defined zones, which are locally called streaks or columns, and more often than not these streaks dip toward the southerly end of the pegmatite. Miners speak of the richer portions of the mica streak or column as mica pockets.

The frequent occurrence of unaltered or only slightly altered granitic material along the footwall, when considered in connection with the predominant albitization of the feldspar and the extensive development of secondary minerals, indicates that originally the pegmatite may have been of granitic structure throughout and that the solutions which caused recrystallization and secondary mineralization did not reach certain portions of the granitic dike at all or not in sufficient quantities to alter the original material greatly. Frank L. Hess, of the United States Bureau of Mines, states that such a history is supported by unquestionable evidence in the case of some pegmatites, and it appears to fit in this instance.

Method of Mining

When the old excavations were cleared in 1925, it was found that the water which had troubled previous operators was almost entirely of surface origin and at no time exceeded 15,000 gallons per day. The water was easily handled by improving the sump and installing a suitable steam pump. The old workings were cleared, caved ground was supported by timbers, and a 24-inch gage track of 30-pound rails was laid down the slope.

The mechanical equipment used in mica mining is seldom of the highest efficiency from an engineering standpoint, as there is usually no way of blocking out or proving ground in advance of mining, and therefore the investment required for machinery incidental to low mining cost is generally not justified.

However, in recent years there has been consistent improvement in portable air compressors and hoists and in drilling equipment, so that reasonably cheap mining can be carried on even in isolated locations and in small pegmatites. With pegmatites from which feldspar as well as mica may be marketed, substantial amounts may be safety invested in equipment.

The mechanical equipment at the No. 10 mine consisted of:

1 - Gasoline powered, portable air compressor of 260-cubic feet per minute capacity.

1 - 25-hp. vertical boiler and single-drum hoist with 3/4-inch cable.

1 - Steam mine pump.

3 - 55-lb. jackhammers, using 7/8-inch hollow steel, chisel bits, columns, and mounts.
1 - Hand shanking device.
1 - Air receiver in mine.
1 - Set of blacksmith tools for drill sharpening.
5 - One-ton mine cars, 24-inch gage.

The blocking cut of mica-bearing ground previous to mining is out of the question in the No. 10 mine. Hence, mining operations were at all times exploration operations as well. The method of mining used was breast stoping following the pegmatite along its slope and using the hanging wall as the principal guide. A good haulage system was of major importance and was always maintained. When a local condition indicated probable mica values below the slope grade, such areas were prospected and mined by breast stoping up the slope and then filled with waste material. This work sometimes reached a depth of 15 feet or more below the track grade. It is reasonably sure that no mica that could be economically recovered was abandoned below the track.

There is evidence that good mica-bearing ground lies beneath the water sump, and when this is mined later, it may lead to more or less extensive workings below present levels. Above the haulage way known mica-bearing areas were passed by, as these can be reached at some future time when the mica and feldspar can be recovered from them by raising the present track on waste material or by sinking another slope at a higher elevation. The general slope of the workings shown in profile in Figure 2 was dictated by the continuity of mica values.

Drill holes were loaded and shot under the supervision of the mine foreman. It frequently happened that the drill would run into block mica, which fact would be at once recognized by the drill runner because of the toughness and elasticity of the material. Generally such holes were abandoned and a change made in placing shots to preserve probable mica values.

Forty per cent dynamite was used throughout. Work was arranged so that the day's shooting could be done at the end of the shift, when the men had left the mine. The shots were fired with a hand-operated magneto. No time was lost waiting for fumes to clear before returning to work. Occasionally shots were fired at the noon hour, conserving the advantages mentioned above. Natural ventilation was relied upon and there was no inconvenience from bad air even at the bottom of the 800-foot slope.

While some alteration, perhaps on account of the intrusion of the pegmatite, has taken place in the wall rocks, the dike material broke cleanly from the walls under the effect of carefully placed shots. It was infrequent that any substantial amount of wall rock was broken down unintentially. At some places after continued blasting it was desirable in the interest of safety to take down slabs of rock that appeared dangerous. Mining was so conducted and conditions were such that no timbering was required. Excavation was not always carried to the footwall, but the ground was explored for values by drilling. Usually two and sometimes three faces were mined at the same time, the crew thus expending its efforts to the best advantage. Acetylene lamps were used by the miners, although electric lighting of the mine would have added somewhat to their convenience and efficiency.

The mine was operated regularly on a 10-hour day shift until 1930, when markets were seriously affected by the readjustment period. The controlling thought was the discovery and recovery of mica. For sufficient reasons, bodies of feldspar that indicated low mica values were passed by. These may be recovered at a later date. The wage rate was a follows:

Operating crew and rates of pay

	Daily wage
1 foreman	$4.50
1 hoist engineer	3.50
1 blacksmith	3.50
2 drill runner	3.00
4 muckers	2.50
1 dumper	2.50
1 team and driver	5.00

About 80 per cent of the block mica recovered was handpicked from the broken rock underground and sacked in burlap bags which were loaded into the mine cars and hoisted to the surface for transportation to the company's mica-processing plants at Spruce Pine. The remainder of the block mica recovered was picked from the dump when the cars of muck were emptied.

The following table summarizes the principal items regarding mining and the yield, size, cost, and value of the mica recovered from this property during a representative 2-year period:

Table 1.- Analysis of mining operations

Rock and spar mined and hoisted	tons	13,655
Rock mined and back filled	do.	800
Total rock and spar mined	do.	14,455
Shots fired		5,113
Holes drilled, total length	feet	25,565
Per ton of rock broken	do.	1.75
Dynamite used (40 per cent)	pounds	9,932
Average charge per hole	do.	1.94
Rock broken per pound of dynamite	tons	1.4
Block mica recovered (Not including mine scrap)	do.	276
Recovered per ton of rock	pounds	38

Table 2.- Analysis of production

Size, inches	Pounds	Total value	Value per pound
CLEAR MICA			
8 x 10	100	-	-
6 x 8	189	-	-
4 x 6	690	-	-
3 x 5	880	-	-
3 x 4	659	-	-
3 x 3	799	-	-
2 x 3	1,886	-	-
2 x 2	1,120	-	-
1½ x 2	1,068	-	-
Total clear mica	7,391	$10,077.31	$1.35
ELECTRIC MICA			
8 x 10	373	-	-
6 x 8	941	-	-
4 x 6	3,577	-	-
3 x 5	3,888	-	-
3 x 4	2,721	-	-
2 x 3	5,602	-	-
2 x 2	14,285	-	-
Total electric mica	33,681	28,588.30	0.85
Total sheet mica	41,072	38,665.61	0.94
Punch mica	156,238	7,761.85	.05
Factory scrap mica	354,776	3,547.76	.01
Total block mica	552,086	49,975.22	0.0905
Value of mine scrap mica and feldspar	-	6,987.14	-
Total value recovered	-	56,762.32	-

Table 3.- Costs

	Total	Per ton of rock mined	Per pound of block mica recovered
Mining labor	$24,455.34	$1.692	$0.0443
Mining supplies, including fuel	9,455.87	.654	.0171
Sheeting costs	5,854.81	.405	.0106
Total cost of mining and sheeting	39,766.02	2.751	0.0720

During the period under consideration mine supplies, including fuel and repairs to mining equipment, averaged 38.6 per cent of the mining labor expense, and 23.8 per cent of the total mining and sheeting cost. Had electric power been available at the mines as it now is, nearly $100 per month in fuel and labor cost could have been saved.

DESCRIPTION OF NO. 21 MINE

The No. 21 mine is situated in Yancey County about 1 mile southwest of Boonford and a little over 3 miles north of the No. 10 mine (see fig. 1). The pegmatite is a pipe of irregular cross section but of fair workable size. It outcrops at the top of a ridge in a section where pegmatites are numerous and pitches nearly due south into the hill, at an angle of about 30° from the horizontal (see fig. 3).

The outcrop of the pipe was weathered and invited exploration many years ago. Mica of clear quality in well-formed crystals was found in paying quantities and the outcrop was opened for a distance of about 60 feet along the pitch. A shaft approximately 30 feet deep with some mining around the bottom represented the last work done years ago and previous to the acquisition of the property by the Spruce Pine Mica Co.

After some testing of the pegmatite it was deemed worth while to open the old workings and to install a good haulage system and suitable pumps to remove the small quantity of water which had to be handled. A slope was driven carrying a 24-inch gage track, and a water sump and pump were installed at the foot of the shaft (see fig. 3).

The rock enclosing the pegmatite is a mica gneiss which shows alteration toward schist along the contacts. The walls are more irregular than usual in this section. The pipe for 200 feet or more from the old shaft is heart-shaped in cross section, apexing in the footwall with the major mica values found near the apex. The irregular shape of the pipe necessitated breaking down considerable wall rock, but no timbering was required. The minerals composing the pegmatite are not well segregated or coarsely crystallized, and as a result there is no marketable feldpar. Accessory minerals are few and of no interest.

The operating equipment consisted of a 7 by 6 inch portable air compressor of sufficient capacity to run a jackhammer or a forge and mine pump as needed, one gasoline-powered hoist, mine cars, drilling equipment, blacksmith tools, etc.

In mining this pegmatite the first consideration was to maintain a haulage way on an even grade. Fortunately, the general form and trend of the pipe were favorable for this. All probable values below the track level were recovered as sinking progressed. Some material on the sides and a considerable quantity overhead was left to be broken and to have the mica recovered from it on retreat, when the waste could be left in the mine. No well-defined or continuous quartz masses so often present in the dikes of the region occur in this pegmatite, but rather the quartz is disseminated throughout the mass, which makes the spar valueless. The average depth of holes drilled for blasting was 5 feet; 40 per cent dynamite was used and shots were fired with electric detonators and a blasting machine. Rounds were usually fired at noon or at the end of the shift, when all men were outside. The mine was operated on a 10-hour day shift.

The working force consisted of a foreman at $4 per day; hoist man and blacksmith combined at $3.50; 1 drill runner at $3; two muckers and one dumper at $2.50, making a total labor cost of $18 per day, without team service. The expense of fuel and other supplies averaged $6.50 per day.

The following tables summarize the principal items regarding mining and the yield, size, cost, and value of the mica recovered from this property during a representative 1-year period:

Table 4.- Analysis of mining operations

Rock mined and hoisted	tons	4,530
Holes drilled, total length	feet	9,290
Per ton of rock broken	do.	2.05
Shots fired		2,066
Dynamite used (40 per cent)	pounds	3.814
Average charge per hole	do.	1.84
Rock broken per pound of dynamite	tons	1.2
Block mica recovered	do.	107
Recovered per ton of rock	pounds	47

Table 5.- Analysis of production

Size, inches	Pounds	Total value	Value per pound
CLEAR MICA			
4 x 6	18	-	-
3 x 5	95	-	-
3 x 4	137	-	-
3 x 3	343	-	-
2 x 3	1,702	-	-
2 x 2	1,641	-	-
$1\frac{1}{2}$ x 2	8,946	-	-
Total clear mica	12,882	$5,391.60	$0.42
ELECTRIC MICA			
3 x 5	6	-	-
3 x 4	12	-	-
3 x 3	27	-	-
2 x 3	185	-	-
2 x 2	447	-	-
Total electric mica	677	251.80	0.365
Total sheet mica	13,559	5,643.40	.417
Punch mica	106,849	5,344.95	.05
Scrap mica	93,833	562.99	.006
Total block mica	214,241	11,551.34	.0539

Table 6.- Costs

	Total	Per ton of rock mined	Per pound of block mica recovered
Mining labor........	$6,519.07	$1.439	$0.0304
Mining supplies (including fuel) ..	1,673.05	.369	.0078
Sheeting costs..........	1,922.90	.424	.0090
Total cost of mining	10,115.02	2.232	0.0472

COMPARISON OF PRODUCTION OF THE NO. 10 AND NO. 21 MINES

Many interesting facts are discovered by comparative analysis of Tables 1 and 4, which give production data for No. 10 and No. 21 mines, respectively.

By dividing the total value of the block mica recovered at the No. 10 mine by the tons of rock broken, the value of the rock for mica alone is found to be $3.46 per ton. If the by-products, feldspar and mine scrap mica, are included, the recoverable value of the rock is $3.93 per ton.

In No. 21 mine where there is no merchantable feldspar or mine scrap mica, the recoverable value in the rock was $2.55 per ton. These values show that even when a pegmatite proves rich or at least rich enough to be worked at a profit, the value per ton of rock considered as ore is low.

A similar analysis of Tables 3 and 6 shows that the mining cost (labor and supplies) per ton of rock was $2.35 and $1.81, respectively, in No. 10 and No. 21 mines; these figures when added to the sheeting costs give $2.75 and $2.23 per ton as the cost of mining and sheeting the block mica recovered.

The mining costs given above are high and plainly indicate the need of more efficient mining methods. It must be remembered, however, that there seems to be no way to prove mica values in advance of mining and that the initial investment required for the installation of machinery for low operating cost is not warranted except in unusual cases. Investment in more efficient equipment might be justified if a group of mines were operated which could be worked out one after the other and if each individual pegmatite were mined rapidly.

It will be noted that there is a great difference in both the size and quality of the mica produced by the two mines. At No. 10 mine, 18 per cent of the sheet mica was clear and 82 per cent was electrical or stained mica. At No. 21 mine, 95 per cent was clear and only 5 per cent of the mica was of electrical grade. At No. 10 mine, 24 per cent of the sheet mica was in the 3 by 5 inch class or larger, while at No. 21 mine, less than 1 per cent went into these larger classes. Finally the average value of the block mica, in sheeted or prepared form, was $181 per ton at No. 10 mine, whereas at No. 21 mine it was $108 per ton. Although the mica from No. 21 mine is not as desirable from the miner's point of view as that from No. 10 mine, its average size and quality are such that the manufacturer finds it eminently suited to present-day demands. It can be worked into highly processed forms and patterns which yield favorable sale values in spite of requiring large expenditures for factory labor as compared with the cost of the crude mica.

SAFETY WORK

The number of men employed at any one of the company's mines was not large enough to make the operation of a safety committee feasible. The work was so completely under the

eyes of the general foreman, that he was made responsible for the maintenance of safe conditions and methods.

The general foreman, the shift foreman, and some of the men had received instruction in safety and first aid from a Bureau of Mines instructor. The importance of safety was stressed at all times, and no opportunity was neglected to make practical application of safety rules and precautions. Machinery was guarded; safe walk ways, ladders, and handrails were installed; and general cleanliness was maintained. The foreman periodically inspected the walls and roof for loose rock, which when found was promptly taken down. Sometimes temporary posts were set to secure fractured or seamed masses until they could be removed.

During the six years in which the company has been engaged in mining, the cost to the company of lost-time accidents has amounted to less than $100 per year. However, the liability insurance rate for mica mining, under the present workman's compensation law in North Carolina is $6.97 per $100 of pay roll. This rate is so high that it is obviously a deterrent to mica mining, but the rate may be susceptible to some reduction if mine operators, with the aid of engineers and inspectors generally, adopt and enforce rigid safety regulations.

DIELECTRIC TESTS OF MICA FROM NO. 10 AND NO. 21 MINES

The following table shows the results of dielectric tests on various thicknesses and grades of mica from the No. 10 and No. 21 mines as compared with similar puncture tests on clear India ruby mica. These tests were made by the Carolina Power and Light Co. in its laboratory at Asheville, N. C., on the latest-type Westinghouse voltage tester using 60-cycle alternating current and 1-inch disk electrodes immersed in oil.

Results of dielectric tests on mica samples by origin and type

Domestic mica						Foreign mica	
No. 10 mine, stained		No. 10 mine, clear		No. 21 mine, clear		India, clear ruby	
Mils	Volts	Mils	Volts	Mils	Volts	Mils	Volts
2	1,900	-	-	-	-	-	-
2.3	2,044	-	-	-	-	-	-
2.5	1,880	2.7	1,666	-	-	2.7	2,074
-	-	3	1,733	3	2,333	3	1,967
3.5	1,809	3	1,800	3.3	2,273	-	-
3.6[1]	2,038	-	-	3.8	2,053	-	-
4.16[1]	2,115	4	1,625	-	-	-	-
4.75[1]	1,716	5	1,460	-	-	-	-
5.42[1]	1,621	-	-	-	-	5.2	1,692
-	-	-	-	5.3	1,623	5.5	1,645
6	1,423	-	-	-	-	-	-
6	1,250	6	1,183	-	-	-	-
6.1	1,352	6.2	1,306	-	-	-	-
6.8	1,353	-	-	6.8	1,558	-	-
7	1,221	7	1,430	-	-	7	1,343
7.5	1,323	-	-	7.4	1,473	7	1,357
-	-	8.8	1,320	-	-	-	-
10	1,070	10	1,160	10	1,350	10	1,020
18	878	-	-	-	-	10	1,120
-	-	-	-	52	573	-	-

1 - Average of three or more tests.

Note: Mils show the thickness of the samples tested, thousandths of an inch.

Volts show breakdown voltage per mil of thickness of the sample tested.

The strictly comparable results of the tests clearly show that the dielectric strength of the stained mica from the No. 10 mine averages quite as high in commercial thicknesses as that of clear mica.

For more complete information on this subject, as well as on other electrical properties of foreign and domestic micas, the reader is referred to Research Paper 347, Volume 7, August, 1931, issued by the United States Bureau of Standards.

Comparison of domestic and India classifications of mica according to size

Domestic classification		India classification	
Class, size, in inches	Rectangular area of sound mica, square inches	Grade	Area of sound mica, square inches
Punch	1 - 3	No. 6	1 to 3
1½ x 2	3 to 4	)	
		) No. 5	3 to 6
2 x 2	4 to 6	)	
2 x 3	6 to 9	)	
		) No. 4	6 to 10
3 x 3	9 to 12	)	
		) No. 4	10 to 15
3 x 4	12 to 15	)	
3 x 5	15 to 24	No. 2	15 to 24
		) No. 1	24 to 36
4 x 6	24 to 48	)	
		) A - 1	36 to 48
6 x 8	48 to 60	)	
		)Specials	48 plus
Specials	60 plus	)	

Note: The domestic classification permits the presence of some imperfect material outside the sound rectangular areas. However, domestic grades are readily converted to India grades by knife trimming.

The mica productions shown in Tables 2 and 5 are classified for size in accordance with domestic standards In this form the material is taken to the factory. Further classification according to quality is then necessary to realize the maximum value in finished products. When specifications are received, the factory organization must select the particular size and quality of sheet from which to process the desired patterns. It must be remembered that the price per pound of mica of any quality increases rapidly with the size, so that in processing mica it is important to use the smallest-sized sheet possible to produce the required pattern.

In domestic practice all loose material is broken from the edges of the rifted sheet mica with the thumb (thumb trim) or is roughly trimmed off with a knife (knife trim), so that as little as possible of the sound material is removed. The resulting product does not compare favorably in appearance with imported mica and for that reason, perhaps, domestic mica has been discredited by some users who buy their supply in uncut form. The domestic practice is sound, however, as the processor or manufacturer can cut or die the largest pattern possible from this rough-trimmed sheet and will have left the maximum-sized trimmings from which to process smaller patterns. It would be a waste of labor and material to trim closely the rifted sheet mica at the mines. If a customer buys uncut mica, it can be closely trimmed or selected to cut the pattern refined with minimum loss. However, importers of mica are right in demanding closely trimmed stock, as it would be unprofitable to pay transportation charges and duty on inferior material.

MICA MANUFACTURING

A brief description of the manufacture or processing of mica products and their ultimate use should be of interest. Mica is an essential material in many applications, as evidenced by the fact that equal efficiency in numerous important uses can not be obtained from any other material, regardless of cost.

Mica is processed or manufactured in three general forms for use in the industries. These three forms are ground mica, mica plate, and sheet and punch mica cut or punched with dies to particular shapes and dimensions.

Ground mica may be either wet or dry ground and is produced from scrap recovered in mining (mine scrap) or from mica processing (factory scrap). Wet-ground mica is usually prepared in chaser mills having wooden rollers and floors, although some is being ground in pebble mills. It is in demand for certain decorative purposes such as in the manufacture of wall paper, in making automobile tires, and for various other uses. Dry-ground mica, a much cheaper product, is generally ground in hammer or rod mills, and finds its principal use in coatings for prepared roofings.

Mica plate, or built-up mica, of various types and qualities, is made from mica splittings of 1 to 6 square inches area and usually about 0.001 inch thick. These splittings are built up or pasted with some binder, shellac, glyptal, etc., into sheets as much as 36 inches wide, of various lengths, and of any thickness desired. The amount of binder used may vary from 2 to 7 per cent of the total weight. The plate is heat-treated under pressure with steam and finished to specified thickness.

The production of mica splittings, several million pounds of which are consumed annually in the United States, is carried on principally in India and to a lesser extent in Madagascar, in both of which countries extremely low wages prevail. The production of splittings has been and still is largely a hand operation, although phlogopite mica in particular is being successfully split by machine in the United States.

We are mainly concerned here with the processing of sheet mica, split or rifted from the rough blocks or crystals taken from the mines, into finished shapes and forms for electric or heat insulation, glazing, and many other uses. These finished sizes range from 1¼ inch in diameter up to patterns 18 inches or even more in linear dimensions and are of simple or complex forms.

The processing of mica into finished forms and shapes, some simple, such as round washers or patterns with straight edges only, and others of intricate design must be done by skilled operatives who must not only be able to use the requisite tools adeptly, but must also thoroughly know the mica which they are handling.

Washers are cut on power presses, with simple dies. Plain forms with no reentering angles are generally best cut with special bench shears. Complicated shapes are necessarily produced with ingeniously designed and accurately finished, hardened steel dies set in power presses. Splitting mica to specified thicknesses is always a tedious hand operation. A variation tolerance in dimensions of a mica pattern of 0.001 inch is costly to attain but commonly demanded.

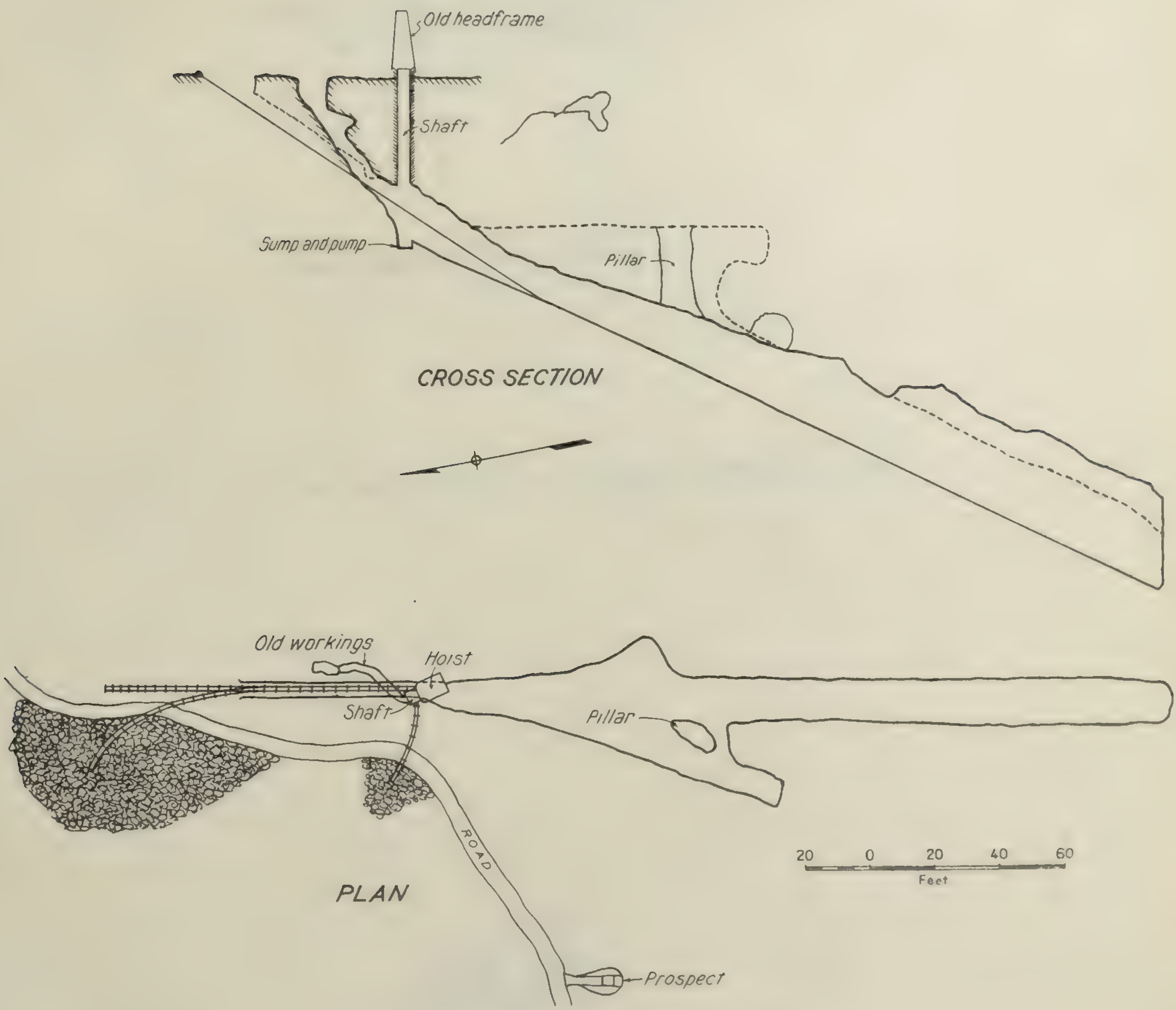

Figure 3.-Plan and cross section of Spruce Pine No.21 mine

DEPARTMENT OF COMMERCE

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FALLS OF ROOF AND COAL IN WASHINGTON MINES

BY

S. H. ASH

I.C.6617
May, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

FALLS OF ROOF AND COAL IN WASHINGTON MINES[1]

By S. H. Ash[2]

ROOF HAZARDS

The coal mines of Washington during the period 1925-1929 had a high index rating (157) due to falls of roof and coal as compared with the average of the coal-producing States of the United States (100), and had even higher rates during preceding 5-year periods, but fortunately the rate for the five years 1926-1930, inclusive, was but 98. The high rate which has obtained until just lately is largely due to roof conditions which undoubtedly in many cases are worse than the average; moreover, there are extra hazardous situations due to bad roof conditions in coal beds that lie on extra heavy dips, and in some instances with extra thick cover. Even in the so-called lightly dipping beds the roof is not of hard massive rock, but generally of coal, bone or shales separated by clay or coal slips. In some places, as in the Roslyn field, the bed carries as its immediate roof a cap rock varying from 4 to 48 inches in thickness. When first opened the roof has the appearance of being good, but unless taken down or timbered, it ultimately falls and causes most of the accidents in the State due to falls of roof.

In the steeply dipping beds both walls are generally bad, and so far as timbering is concerned the roof and floor must be treated almost alike. This is due to possibilities of slides of coal or rock down the pitch, which until 1917 constituted about 12 per cent of the total number of accidents, while falls of roof and coal for the entire State constituted about 28 per cent of the total. From the figures given it can be deduced that 16 per cent of the accidents were due to falls of roof and face coal and 12 per cent to slides, and in at least some instances the sliding material came from the floor. The prevention of such accidents, which are classed as falls or caves, depends in one way or another on factors which involve timbering.

In addition to accidents due in some manner to timbering, or to lack of it, until 1917, 4 per cent of the total number of accidents was from falling down chutes. This hazard also is more or less directly connected with timbering practice; batteries may give way, props fall out, ladders slip, and because of lack of landings men fall considerable distances if they slip.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6617."

2 District engineer, U. S. Bureau of Mines Safety Station, Berkeley, Calif.

Since 1917 the percentage of fatal accidents due to falls of roof and coal, which includes slides, has increased at a rapid rate, and in 1929 they constituted 54.5 per cent of the total number of accidents occurring in the coal mines of Washington. The reasons for this increase are that accidents due to the causes mentioned are about the same in number, and attempts at their prevention are more or less at a standstill, while preventive measures have been taken in other directions, notably in gas explosion hazards. The total number of accidents has been decreasing slowly, which necessarily raises the percentage due to falls, when the number due to this cause remains stationary. In Washington, as elsewhere, there is a fertile field for accident-prevention work to eliminate accidents caused by falls of roof and coal.

In the pitching bed districts no advance can be made without timber in most gangways and other narrow work driven on the strike, because roof conditions require temporary timbering with posts or forepoling until a set can be put in place. Inasmuch as timbering is purely a manual operation, mechanical loaders effect a substantial saving in time taken up by loading and should tend to decrease the hazards caused by falls because there should be less time exposure under roof that can not be permanently timbered until the coal or waste is cleaned up, and slabs have less time in which to work loose. However, because of the noise and crowding of the work this safety factor is often nullified; moreover, some types of loading machines require so much room that an undue amount of untimbered space must be left to allow the machine to operate. Strict supervision and frequent inspection are more necessary whether loading is done mechanically or by hand.

Roof Testing Not Done

Probably one of the principal failings of the individual workmen that tends to keep up the high rate of fatalities from falls of roof is indifference toward testing the roof, especially in pillar operations. Even when testing is done, the method employed is merely to tap the roof with a pick instead of also feeling by the free hand for vibration of the roof. In high coal too much dependence is placed on looking at the roof rather than testing it. In testing high roof a metal pipe should be used; and whether in high or in low coal, persons should keep from under untimbered roof while testing.

In working thick beds with a strong roof that does not cave in large extracted areas there is a constant danger in any bed and on any dip because of air compression or, as it is usually expressed, because of the force of the wind developed when the roof caves in a body. The danger is increased immeasurably in a pitching bed opened by narrow chutes, not only from the rush of loose material, but on account of the small passageways; a terrific force is developed by the air which escapes under pressure through the narrow chutes. The remedy for such occurrences is to break the roof in some manner or run the loose material available to fill the area immediately above the pillar face.

The lack of discipline and failure to enforce safety rules is the indirect if not the direct cause of many accidents, but especially for many accidents from falls of roof or face coal. The following report by the State mine inspector of an accident is typical of many accidents that can usually be assigned to failure on the part of the supervisor and workman:

> January 29, 1920.____________, a miner working in a chute, was caught under a falling rock, injuring him internally, causing death. There were no props within a radius of seven feet where ____________ was working. In cases like this it is the duty of the mine foreman to insist on miners' properly timbering their working places.

Bumps

Bumps or bounces have occurred in two districts of the Washington coal mines where the cover ranges from 1,500 to 2,800 feet. In the King County district[3] some of these bumps resulted in fatal accidents as well as property damage. Their frequency was a contributing reason for abandoning one large mine. The same experience was met in the Carbonado field and finally resulted in the closing of the mine affected.

In both districts the coal beds had a strong roof of massive sandstone and were almost identical in thickness, which varied from $4\frac{1}{2}$ to 6 feet. Their physical characteristics were similar and both were in a region of faults.

The occurrences of the bumps were also similar and were generally preceded by periods of quietness. Small bumps which occurred regularly were interpreted by the miners to indicate that everything was normal; if these slight bumps were not felt, a dangerous one could be reasonably expected.

In many ways the physical effects of the bumps were the same - namely, a closing in of the workings, cave of loose material, shattering of pillars, and breakage of timber. However, in the Carbonado district the coal bed was gassy and the bumps released large volumes of methane to such an extent that men were suffocated. This was not true in the King County district.

The cause of the bumps was the movement of the main roof to such a degree that the force transmitted through the pillars crushed them and relieved itself by causing the floor to heave. This movement in turn would break the timber, hurl the workman or material against the roof or sides of the pillars, close the crosscuts, liberate large volumes of gas in the Carbonado area, and fill the places with loose coal; if anyone happened to be under a loose piece of roof or in the way of flying material a fatality often resulted.

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3 Evans, G. W., Coal-Mining Problems in the State of Washington: Bull. 190, Bureau of Mines, 1924, p. 32.

At one mine in the King County district 15 men were killed between December, 1920, and June, 1924, 13 by falls of rock, 1 by being hurled against the roof, and 1 by being struck by a broken prop.

The mine inspectors' reports in numerous cases record hazards from bumps.

It is obvious that bumps are a result of pressure, generally from the roof, and that if some method can be devised to control this pressure there will not be any bumps. It has been observed that certain beds at the same depth in the same districts where bumps occur, but having walls that tend to swell and fill the workings, do not present any serious hazard from bumps. If a bed has two benches, or if two beds are close enough to allow the bottom one to cave up to the second, such as is the case in the Washington districts, where the bumps occur, it is probable that when the parting between the two benches is firm and thick enough, and one or both of the walls at the top bench are likely to heave or sag to such a degree as to allow the main roof to break over the waste workings in the top bed or bench, an excellent method is offered of avoiding bumps in the workings of either bed or bench in deep mines when the walls of the lower bench are firm. If the lift is made such at a longwall method[4] is used, the main roof will break, but owing to the filling of the top bench workings with waste no serious consequences result. The bottom bench can then be worked and caves obtained as desired. However, if this is not done the top bench is generally lost; the lower bench workings remain open for a considerable area, and when the main roof does let go, a tremendous pressure is instantly thrown on the adjacent workings and a serious bump results, crushing the pillars, breaking the timber, and at times caving-in that section of the mine. This is especially true in a region of faulting. If the coal makes much gas, a large quantity of it is often liberated from the crushed pillars at the same time. If the caving of the roof can be regulated, usually the bottom will not give serious trouble from bumps.

A retreat system of mining will generally eliminate bumps where the cover is thick, even when advance systems are practiced for a considerable depth, and will minimize them at any depth. Maintenance of a definite pillar line is the first essential for the prevention of bumps under any system. Coal mining in Washington does not yield a profit large enough to permit the use of a back-filling system, which would tend to eliminate bumps to economic depths.

ACCIDENTS FROM FALLS OF ROOF AND COAL

The chief danger to which coal miners are exposed is that of falling roof and coal. In the industry as a whole accidents from this hazard comprise about 48 per cent (57 per cent in 1931) of all accidental deaths in the mines and about 33 per cent of all nonfatal injuries that are serious enough to prevent an employee from reporting for duty on the day following the accident. In 1931, 824 men were killed by falls of roof and coal out of a total of 1,43[illegible] fatalities at the coal mines in the United States.[5]

4 Ash, S. H., Systems of Coal Mining in Western Washington: Trans. Amer. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 833-873.

5 Adams, W. W., and Chenoweth, L., Coal-Mine Fatalities in December, 1931: Health and Safety Statistical Surveys, C.M.F. 6, Bureau of Mines, Jan., 19[illegible]

In Washington,[6] as elsewhere, most of the mine accidents, both fatal and nonfatal, occur at or near the face and are due to falls of roof or coal. Apparently, these accidents are not considered avoidable by many companies, probably because their prevention frequently lies in the hands of the individual workman; however, this fact does not by any means absolve the management of ultimate responsibility. No one can question the fact that if the roof or face had been properly supported, in nearly every instance there would not have been any accident from a fall of roof or coal.

Table 1 shows the number of fatal accidents which occurred during 1905 to 1931 in Washington coal mines due to falls, and gas, or gas and coal-dust explosions.

Table 1.- Fatal accidents in Washington coal mines due to falls of rock and coal and explosions, 1905-1931

Year	Total fatalities, all causes	Falls of roof		Falls of face or pillar coal		Falls of rock or coal		Explosions of gas or dust	
		Number	Per cent	Number	Per cent	Number	Per cent	Number	Per cent
1905-1916[1]	325	-	-	-	-	92	28.30	84	25.84
1917-1927[1]	246	74	30.08	13	5.28	87	35.36	22	13.00
1928 [1]	9	4	44.44	1	11.11	5	55.55	1	11.11
1929 [1]	11	5	45.45	1	9.09	6	54.54	0	00.00
1930 [2]	23	1	4.35	1	4.35	2	8.70	17	73.91
1931 [2]	8	1	12.50	3	37.50	4	50.00	0	0.00

1 Compiled from reports of State Mine Inspectors.
2 Compiled from reports of the U. S. Bureau of Mines; those for 1931 are subject to revision.

It is interesting to note the decline in accidents due to explosions and the slight increase of accidents due to falls. This is also a result of strict enforcement of the new safety regulations relating to gassy mines, and perhaps less strict enforcement of the sections of the mining law relating to timbering. The reason for this is traceable to the fact that, as in times past, the miner is largely responsible for setting timber.

Statistics on Washington show that the nonfatal accidents due to falls also form about 37 per cent of the total, or the largest percentage of nonfatal accidents from any one cause. The time loss from falls alone averages about 24 days to each accident, and constitutes about 35 per cent of the total time loss due to injuries.

6 Ash, S. H., Systematic Timbering Rules at the Washington Coal Mines: Inf. Cir. 6316, Bureau of Mines, 1930, 8 pp.

PREVENTION OF ACCIDENTS DUE TO FALLS

In any scheme[7] for protection against roof hazards, there must be:

1. Some definite system of setting timber.
2. Diagrams showing how and where to set the timber.
3. Instructions given the miners and supervisors.
4. Supervision and regular inspection.
5. A record kept of any neglect to set timber.
6. A penalty for neglect to observe the timber regulations.

As a rule, accidents from falls of roof or coal are caused by lack of proper roof or face support.[8] With few exceptions this can be blamed on the failure of the individual workman to set timber, whether through carelessness or ignorance, although in a few cases lack of timber has been the contributing cause. Obviously, the remedy is to enforce systematic timbering rules by strict discipline for failure to set timber. Experience over a number of years had proved that under the same working conditions fewer accidents due to falls occur at mines where systematic timbering rules are formulated and are actually enforced. Where systematic timbering rules are made, lack of discipline is the principal cause of accidents from falls of roof, face, or pillar coal. It is rapidly becoming recognized that lack of supervision and discipline is the major cause of laxity in following safe practice and lack of efficiency on the part of workmen around coal mines. Rules are ineffectual if they are not enforced, and men should be disciplined for violation of rules, even if no harm has resulted, rather than punished only after damage has been done.

HEAD PROTECTION

An examination of the accident statistics of Washington mines reveals the outstanding fact that there are an unusual number of nonfatal accidents from being struck on the head by coal, rock, and other material's rolling or falling down chutes or breaking loose through bulkheads on the pitch. Probably the general adoption of head protection is nowhere more needed than in Washington mines because of the steep pitches, coupled with bad roof and high coal in many instances. One of the largest companies has seen fit during the past two years to require head protection, and the results are speaking for themselves. Unfortunately, however, the benefits that are insured by head protection are sometimes conducive, on the part of some men, to taking chances in high-roof places and to expecting immunity far beyond that which such safety equipment provides, or for which it is intended. Head protection is not infallible in preventing serious injuries from material falling from great heights, and the maximum benefits from head protection will result when there is no laxity of ordinary or usual safety practices following the adoption of such safety devices. There is a striking similarity at times with the conditions not infrequently found in gassy mines after the adoption of closed lights. In the latter, if adequate ventilation is neglected because open lights have been eliminated, disaster will also inevitably result.

7 Paul, J. W., Reducing Accidents from Falls of Roof in Coal Mines, Part I - Six Essentials for Mine Roof Support: Inf. Circ. 6225, Bureau of Mines, 1930, 11 pp.

8 See footnote 6.

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ACCIDENT EXPERIENCE AND COST IN PENNSYLVANIA ANTHRACITE AND BITUMINOUS MINES, 1926-1930

BY

W. J. FENE

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

ACCIDENT EXPERIENCE AND COST IN PENNSYLVANIA ANTHRACITE AND BITUMINOUS MINES, 1926-1930[1]

By W. J. Fene[2]

PURPOSE OF THIS REPORT

The prevalence of accidents in the coal-mining industry has prompted a study of the workmen's compensation laws of the various States and the statistics of accident costs, for the purpose of explaining to operators how the compensation laws are regulated and to show the cost of accidents.

The subject of compensation laws and accident costs is by no means new, but the prevalence of coal-mine accidents would indicate that the mine operator in general is not thoroughly conversant with their importance. Unquestionably, compensation laws are a large factor in promoting accident prevention, and it is believed that a greater measure of success would be accomplished in preventing accidents if all mine operators were fully aware of their important bearing on the cost of production.

Many operators seem to be indifferent to accident occurrence; possibly they do not realize the seriousness of the accident hazards that exist in their mines, even after the hazards are brought to their attention. There is no question that material financial advantages are to be gained by the prevention of accidents and it would appear that when these advantages are thoroughly understood, operators would be anxious to eliminate, insofar as is feasible, conditions likely to cause accidents.

The unsettled state of the mineral industry and the irregular operation of mines during the past several years has probably been an important factor in influencing the apparently indifferent attitude of the operator. The chaotic conditions in the mining industry and especially in coal mining are unquestionably a factor that tends to increase accidents, just as mine idleness increases roof hazards and results in deterioration in mechanical, electrical, and haulage equipment, and promotes disorganization of the personnel. During such times greater efforts should be exerted toward the prevention of accidents because of these increased hazards and of the financial advantages brought about by the various savings to be effected when mine accidents are held to a minimum.

ACKNOWLEDGMENTS

This report on accidents in the coal mines of Pennsylvania was made possible largely through the cooperation of William J. Maguire, director, Bureau of Statistics, Pennsylvania Department of Labor and Industry, and Rush N. Hosler, superintendent, Coal Mine Section, Pennsylvania Compensation Rating and Inspection Bureau, who made available their records and extended much assistance and numerous courtesies during this study.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6618."

2 - Associate mining engineer, U. S. Bureau of Mines Safety Station, Pittsburgh, Pa.

PENNSYLVANIA WORKMEN'S COMPENSATION LAW

The Pennsylvania Workmen's Compensation Act was approved on June 2, 1915, and became effective on January 1, 1916. Every employer of labor in the Commonwealth of Pennsylvania, under the provision of the Workmen's Compensation Law, must insure the payment of compensation by carrying compensation insurance in any insurance company authorized to insure such liability in the Commonwealth, or must secure the privilege of operating as a self-insurer. That compensation legislation has been a boon to industrial workers of Pennsylvania is shown by the fact that during the 15-year period in which the law has been in effect, 1,098,075 workers or their families have been beneficiaries; the ratio is approximately one beneficiary for every 3.4 of working population. The direct payments to injured workers or to their dependents have averaged $12,148,814 annually.

PENNSYLVANIA WORKMEN'S COMPENSATION BUREAU AND BOARD

The Pennsylvania Workmen's Compensation Law is elective (or optional) as to compensation and compulsory as to insurance. All workers except farm laborers, domestic servants, and casual employees are covered. Electing employers must insure with the State Insurance Fund or with private casualty companies or provide self-insurance. The State fund is not administered by the Workmen's Compensation Bureau or Board, but by a special Workmen's Insurance Board consisting of the secretary of labor and industry, the insurance commissioner, and the State treasurer. The compensation act is administered by two State departments. The initial administration of the act is under the Bureau of Workmen's Compensation of the Department of Labor and Industry; this bureau has supervision up to the time the case goes to a hearing. In case of a dispute, or when an application for a hearing is made, the matter comes for hearing before one of the district referees. These referees are independent of each other, but are directly responsible to the secretary of labor and industry, and to the Workmen's Compensation Board. If either party is dissatisfied with the decision of the referee, an appeal is taken to the Workmen's Compensation Board, an independent judicial body of which the secretary of labor and industry is an ex-officio member but has no vote in the disposition of cases brought before the board.

The Workmen's Compensation Board consists of three members appointed by the governor, with the advice and consent of the Senate, for a term of five years. The secretary of labor and industry is also appointed by the governor; the secretary is authorized to appoint the referees and other employees of the bureau. The Workmen's Compensation Board is authorized to appoint a secretary and other clerical employees. The administrative expenses of both the Workmen's Compensation Bureau and the Workmen's Compensation Board are paid out of the State treasury from moneys regularly appropriated.

As already stated, neither the Workmen's Compensation Bureau nor the Workmen's Compensation Board has jurisdiction over the State fund, nor have they direct supervision over accident prevention. The latter work for general industry is under the direction of the Department of Labor and Industry, but is performed by the inspection bureau of the department and for the coal-mining industry it is performed by the Department of Mines.

The compensable accident cases are settled by voluntary agreements, which must be approved by the bureau. In disputed cases, either party may file an application for a hearing before a referee. Appeal may be had from the referee's decision to the Workmen's Compensation Board and from the board to the courts.

IMPORTANT FEATURES OF THE COMPENSATION LAW

The following sections of the Pennsylvania Workmen's Compensation Law are of importance to mining men in the handling of compensation cases:

Section 306. Schedule of compensation.

The following schedule of compensation is hereby established for injuries resulting in total disability:

(a) Total disability. For the first five hundred weeks after the seventh day of total disability, sixty-five per centum of the wages of the injured employee as defined in section three hundred and nine; but the compensation shall not be more than fifteen dollars per week nor less than seven dollars per week, and shall not exceed in aggregate the sum of sixty-five hundred dollars: Provided, that, if at any time of injury the employee receives wages of less than seven dollars per week, then he shall receive the full amount of such wages per week as compensation. Nothing in this clause shall require payment of compensation after disability shall cease. Should partial disability be followed by total disability, the period of five hundred weeks mentioned in this clause of this section shall be reduced by the number of weeks during which compensation was paid for such partial disability.

(b) Partial disability. For disability partial in character (except the particular cases mentioned in clause (c), sixty-five per centum of the difference between the wages of the injured employee, as defined in section three hundred and nine, and the earning power of the employee thereafter; but such compensation shall not be more than fifteen dollars per week. This compensation shall be paid during the period of such partial disability; not, however, beyond three hundred weeks after the seventh day of such partial disability. Should total disability be followed by partial disability, the period of three hundred weeks mentioned in this clause shall be reduced by the number of weeks during which compensation was paid for such total disability.

(c) Permanent injuries. For all disability resulting from permanent injuries of the following classes, the compensation shall be exclusively as follows:

For the loss of a hand, sixty-five per centum of wages during one hundred and seventy-five weeks.

For the loss of an arm, sixty-five per centum of wages during two hundred and fifteen weeks.

For the loss of a foot, sixty-five per centum of wages during one hundred and fifty weeks.

For the loss of a leg, sixty-five per centum of wages during two hundred and fifteen weeks.

For the loss of an eye, sixty-five per centum of wages during one hundred and twenty-five weeks.

For the loss of a thumb, sixty-five per centum of wages during sixty weeks.

For the loss of a first finger, commonly called index finger, sixty-five per centum of wages during thirty-five weeks.

For the loss of a second finger, sixty-five per centum of wages during thirty weeks.

For the loss of a third finger, sixty-five per centum of wages during twenty weeks.

For the loss of a fourth finger, commonly called little finger, sixty-five per centum of wages during fifteen weeks.

The loss of the first phalanx of the thumb, or of any finger, shall be considered equivalent to the loss of one-half of such thumb or finger, and shall be compensated at the same rate as for the loss of a thumb or finger, but for one-half of the period provided for the loss of a thumb or finger.

The loss of more than one phalanx of a thumb or finger shall be considered equivalent to the loss of the entire thumb or finger.

For the loss of any two or more such members, or the permanent loss of the use of the hand, arm, foot, leg, or eye, as herein before provided, not constituting total disability, sixty-five per centum of wages during the aggregate of the periods specified for each.

For serious and permanent disfigurement of the head or face of such a character as to produce an unsightly appearance, and such as is not usually incident to the employment, sixty-five per centum of wages not to exceed one hundred and fifty weeks.

Unless the Board shall otherwise determine, the loss of both hands or both arms or both feet or both legs or both eyes shall constitute total disability, to be compensated according to the provisions of clause (a).

Amputation between the elbow and the wrist shall be considered as the equivalent of the loss of a hand, and amputation between the knee and ankle shall be considered as the equivalent of the loss of a foot. Amputation at or above the elbow shall be considered as the loss of an arm, and emputation at or above the knee shall be considered as the loss of a leg. Permanent loss of the use of a hand, arm, foot, leg, eye, finger or thumb, shall be considered as the equivalent of the loss of such hand, arm, foot, leg, eye, finger or thumb.

This compensation shall not be more than fifteen dollars per week nor less than seven dollars per week: Provided, That, if at the time of injury the employee receives wages of less than seven dollars per week then he shall receive the full amount of such wages per week as compensation.

(d) When compensation begins. No compensation shall be allowed for the first seven days after disability begins, except as hereinafter provided in clause (e) of this section.

(e) Surgical and medical services. During the first thirty days after disability begins, the employer shall furnish reasonable surgical and medical services, medicines, and supplies, as and when needed, unless the employee refuses to allow them to be furnished by the employer. The cost of such services, medicines, and supplies shall not exceed one hundred dollars ($100). If the employer shall upon application made to him, refuse to furnish such services, medicines, and supplies, the employee may procure the same and shall receive from the employer the reasonable cost thereof within the above limitations. In addition to the above service, medicines, and supplies, hospital treatment, services, and supplies shall be furnished by the employer for the said period of thirty days. The cost for such hospital treatment, service, and supplies, shall not in any case exceed the prevailing charge in the hospital for like services to other individuals. If the employee shall refuse reasonable surgical, medical, and hospital services, medicines, and supplies, tendered to him by his employer, he shall forfeit all right to compensation for any injury or any increase in his incapacity shown to have resulted from such refusal.

(f) Death of employee. Should the employee die as a result of the injury, the period during which compensation shall be payable to his dependents, under section three hundred and seven of this article, shall be reduced by the period during which compensation was paid to him in his lifetime, under this section of this article. No reduction shall be made for the amount which may have been paid, or contracted to be paid, for medical and hospital services and medicines, nor for the expenses of the last sickness and burial. Should the employee die from some other cause than the injury, the liability for compensation shall cease.

(g) Hernia. Hernia shall be considered as a physical weakness or ailment, which ordinarily develops gradually, and shall not be compensable, unless conclusive proof is offered that the hernia was immediately precipitated by such sudden effort or severe strain that: First, the descent of the hernia immediately followed the cause; second, there was actual pain in the hernial region; third, the above manifestations were of such severity that the same were immediately noticed by the claimant and communicated to the employer, or a representative of the employer, within forty-eight hours after the occurrence of the accident.

Section 307. Compensation in case of death of employee.

In case of death, compensation shall be computed on the following basis, and distributed to the following persons:

1. To child or children. If there be no widow nor widower entitled to compensation, compensation shall be paid to the guardian of the child or children, or, if there be no guardian to such other persons as may be designated by the board as hereinafter provided, as follows:

(a) If there be either one or two children, thirty-three per centum of wages of deceased, but not in excess of seven dollars and fifty cents per week.

(b) If there be three children, forty-four per centum of wages of deceased but not in excess of ten dollars per week.

(c) If there be four children, fifty-five per centum of wages of deceased, but not in excess of twelve dollars and fifty cents per week.

(d) If there be five children, sixty-two and one-half per centum of wages of deceased, but not in excess of fourteen dollars per week.

(e) If there be six or more children, sixty-five per centum of wages of deceased, but not in excess of fifteen dollars per week.

2. To widow or widower, if no children. To the widow or widower, if there be no children, forty-four per centum of wages, but not in excess of ten dollars per week.

3. If one child. To the widow or the widower, if there be one child, fifty-five per centum of wages, but not in excess of twelve and a half dollars per week.

4. If two children. To the widow or widower, if there be two children, sixty-two and one-half per centum of wages, but not in excess of fourteen dollars per week.

4½. If three or more children. To the widow or widower, if there be three or more children, sixty-five per centum of wages, but not in excess of fifteen dollars per week.

5. To father and mother. If there be neither widow, widower, nor children entitled to compensation, then to the father or mother, if dependent to any extent upon the employee at the time of the accident, twenty-five per centum of wages, but not in excess of five dollars per week: Provided, however, That in the case of a minor child who has been contributing to his parents, the depend-

ency of said parents shall be presumed, and: Provided, further, That if the father or mother was totally dependent upon the deceased employee at the time of the accident, the compensation payable to such father or mother shall be forty-five per centum of wages, but not in excess of ten dollars per week.

6. To brothers and sisters. If there be neither widow, widower, children, nor dependent parent, entitled to compensation, then to the brothers and sisters, if actually dependent to any extent upon the decedent for support at the time of his death, fifteen per centum of wages for one brother or sister, and five per centum additional for each additional brother or sister, with a maximum of twenty five per centum, such compensation to be paid to their guardian, or, if there be no guardian, to such other person as may be designated by the board, as hereinafter provided, as follows:

7. Expense of burial. Whether or not there be dependents as aforesaid, the reasonable expense of burial, not exceeding one hundred and fifty dollars, which shall be paid by the employer or insurer directly to the undertaker (without deduction of any amounts theretofore paid for compensation or for medical expense).

COAL-MINE COMPENSATION INSURANCE RATES

The insured coal mines of Pennsylvania are rated for compensation insurance by the Coal Mine Section of the Pennsylvania Rating and Inspection Bureau which was established by State legislative action.

HOW PREMIUM RATES ARE COMPUTED

The cost of insurance is expressed in terms of a rate, which is the cost per unit of exposure, the rate representing the price per $100 of pay-roll expenditure. The pay-roll basis is used because compensation payments are computed on the wages of injured employees and the pay-roll indicates the extent of the insured's operation and the consequent exposure to accidents.

Premium rates are determined by combining (1) the fixed minimum rate, (2) experience rate, and (3) schedule rate.

(1) Fixed minimum rate.- This rate is determined by taking the average accident experience of all insured mines in the State. To the fixed minimum rate is then added the experience rate and the schedule rate, to give the premium rate. This fixed minimum may be changed from year to year, depending on the aggregate experience of the State and the average charges developed on all mines.

(2) Experience rate.- Experience rating covers all the hazards of a mine, and deals with actual loss experience of the individual mine. It brings into the rate determination the evidence contained in the loss history of the mine, and to some extent it parallels the schedule, but it shows a somewhat closer approximation of the true hazard of the individual mine.

Experience rates are determined by using as a basis the accident experience of the mine during the previous five years.

Every coal mine which has developed an average annual pay-roll of $25,000 during the experience period is subject to experience rating. The experience period is five full policy years next preceding the date of experience rate calculation, or so many full policy years as the operator has been insured under the Workmen's Compensation Act, but not less than two full policy years. For the operator who has not been insured and submits his own experience, the experience period is the five calendar years next preceding the date of the experience rate calculation.

The aggregate pay-rolls and losses of an operator are treated as a unit in calculating the experience rate. The operator's experience includes the pay-roll and losses of each and every mine operated by such operator in Pennsylvania during the experience period.

In calculating losses for experience rating plan, no consideration is taken of loss arising out of any catastrophe. For this purpose, a catastrophe is defined as a single accident which causes five or more deaths. A catastrophe rate of 0.15 cent for bituminous mines and 0.06 cent for anthracite mines is made to cover losses from this cause.

(3) Schedule rate.- Every mine insured under a workmen's compensation policy which has an estimated annual pay-roll of at least $10,000, or in which 10 or more men are employed shall be specifically rated in accordance with the schedule.

COMPARISON OF ANTHRACITE AND BITUMINOUS PREMIUM CHARGES BY ITEMS - 1931

Tables 1~~, 2, and 3~~ give the results of coal-mine schedules and experience rating under the 1931 schedule, as promulgated by the Coal Mine Section of the Pennsylvania Compensation Rating and Inspection Bureau. The charges shown are made for mines other than those which are self-insured, and represent approximately 50 per cent of the bituminous tonnage mined and 10 per cent of the anthracite tonnage mined.

The rates are based on schedule ratings and average experience over a period of the preceding five years.

In Table 1, showing the schedule premium charge, the column "Premium charges" represents the total premium to be paid for the different charge items. "Rate value" is the charge made in the schedule for different hazards; it is the average charge made after credit has been deducted from the rate value. It is shown that item VI, "Roof and pitch of bed" represents the greatest per cent of all charges in both anthracite and bituminous mining. There is considerable difference in the rate value between anthracite and bituminous, being $3.58 and $4.50, respectively, which does not necessarily indicate the relative hazard of this item in mining the two kinds of coal. Item IV, "Haulage underground," represents the second greatest percentage of all charges, being slightly higher for anthracite mines than for bituminous mines. It is observed that item VII, "Explosives," represents 17.4 per cent of all charges in anthracite mines, while in bituminous mines explosives represent only 4.6 per cent of all charges. The rate charge for explosives for anthracite mines is 0.59 and for bituminous mines the rate is 0.22. The large differential shown by this charge in the two kinds of mines may be accounted for by the reason that in anthracite mining a greater quantity of explosives is required per ton of coal produced than in bituminous mining; and as the exposure hazard materially increases with increased quantity of explosives used, the rate value would therefore increase.

The credit items referred to include safety organization, inspection service, and rock-dusting in bituminous mines.

The minimum anthracite rate for 1931 was $3.15 per $100 of pay-roll as against $3.85 for bituminous mines.

Table 1.– Schedule premium charges, showing value by item classes--1931

Classes 1	Premium charges 2	Per cent of all charges 3	Rate value 4	Rate charge 5
ANTHRACITE				
Net charges	$89,735	95.0	–	$0.867
Less credit items	4,685	5.0	–	.045
All classes	94,420	100.0	–	.912
II. Surface hazards	3,923	4.1	$0.30	0.038
IV. Haulage underground	22,689	24.1	.78	.219
V. Electricity underground	1,282	1.3	.15	.013
VI. Roof and pitch of bed	43,570	46.2	3.58	.420
VII. Explosives	16,440	17.4	.59	.159
VIII. Gas and fires	2,923	3.1	.75	.028
IX. Miscellaneous hazards	3,593	3.8	.26	.035
BITUMINOUS				
Net charges	$805,673	90.8	–	$0.995
Less credit items	81,915	9.2	–	.101
All classes	887,588	100.0	–	1.096
II. Surface hazards	52,914	6.0	$0.32	0.065
IV. Haulage underground	181,896	20.5	1.10	.225
V. Electricity underground	26,757	3.0	.12	.033
VI. Roof and falls	443,979	50.0	4.50	.549
VII. Explosives	41,370	4.6	.22	.051
VIII. Gas, dust, and fires	88,476	10.0	.61	.109
IX. Miscellaneous hazards	52,196	5.9	.35	.064

Schedule Rates by Rate Groups

Table 2 shows the schedule rates by rate groups for anthracite and bituminous mines. The average rate for anthracite mines is $4.03 and $4.84 for bituminous mines, and the maximum charge for anthracite is $5.17 as against $7.21 for bituminous. These schedule rates are based on 65 anthracite mines and 706 bituminous mines. During 1930, 248 anthracite and 1336 bituminous mines operated in Pennsylvania. It seems probable that if the rates were based on a greater number of anthracite mines the difference between anthracite and bituminous rates would be considerably smaller.

Table 2.- Schedule rates by rate groups—1931

Rate group 1	Mines 2	Pay-roll 3	Schedule adjusted premium 4	Premium charges 5	Average rate 6
ANTHRACITE					
All rates	65	$10,366,500	$417,625	$94,420	$4.03
Under $3.50	2	673,500	23,505	3,973	3.49
$3.50 to $3.75	5	1,405,500	50,934	9,401	3.62
3.75 to 4.00	19	2,329,500	90,098	16,717	3.87
4.00 to 4.25	18	3,280,500	135,127	32,051	4.12
4.25 to 4.50	15	2,191,500	95,082	26,050	4.34
4.50 to 4.75	4	370,500	16,951	3,938	4.58
4.75 to 5.00	1	19,500	965	351	4.95
Over $5.00	1	96,000	4,963	1,939	5.17
BITUMINOUS					
All rates	706	$80,952,000	$3,920,797	$887,588	$4.84
Under $4.00	27	4,627,500	180,574	13,984	3.90
$4.00 to $4.25	52	11,848,500	491,407	62,284	4.15
4.25 to 4.50	78	11,949,000	521,563	80,346	4.36
4.50 to 4.75	90	9,186,000	423,501	75,769	4.61
4.75 to 5.00	100	13,231,500	644,057	144,727	4.87
5.00 to 5.25	83	10,036,500	512,238	134,412	5.10
5.25 to 5.50	94	6,538,500	351,692	100,891	5.38
5.50 to 5.75	77	6,768,000	378,718	118,641	5.60
5.75 to 6.00	48	3,454,500	203,175	70,176	5.88
6.00 to 6.25	26	1,057,500	64,995	24,279	6.15
6.25 to 6.50	15	639,000	40,571	15,971	6.35
6.50 to 7.00	14	1,485,000	98,903	41,730	6.66
Over $7.00	2	130,500	9,403	4,378	7.21

Table 3.- Bituminous schedule rates by counties

County	Mines	Pay-roll	Schedule adjusted premium	Premium charges	Average rate
1	2	3	4	5	6
All counties	706	$80,952,000	$3,920,797	$887,588	$4.84
Allegheny	77	9,699,000	453,942	94,941	4.68
Armstrong	20	1,957,500	95,037	19,671	4.86
Beaver	8	301,500	16,672	5,063	5.53
Bedford[1]	10	447,000	23,337	6,128	5.22
Blair	9	633,000	28,180	4,637	4.45
Bradford	1	27,000	1,480	440	5.48
Butler	33	2,431,500	127,549	33,938	5.25
Cambria	114	16,851,000	749,540	122,467	4.45
Cameron	-		-	-	-
Centre	22	1,404,000	71,129	17,129	5.07
Clarion	19	1,050,000	56,509	16,353	5.38
Clearfield	80	4,825,500	235,707	51,154	4.88
Clinton	4	264,000	12,355	2,191	4.68
Elk	8	241,500	12,605	3,328	5.22
Fayette	51	6,600,000	334,719	90,523	5.07
Greene	11	4,716,000	220,771	50,347	4.68
Huntingdon[1]	6	1,267,500	65,777	16,977	5.19
Indiana	29	2,586,000	118,640	21,297	4.59
Jefferson	16	945,000	45,291	9,900	4.79
Lawrence	3	126,000	7,308	2,458	5.80
Lycoming	1	109,500	5,825	1,610	5.32
McKean	-	-	-	-	-
Mercer	6	228,000	12,567	3,786	5.51
Somerset	83	8,472,000	393,708	75,185	4.65
Tioga	6	282,000	14,037	3,181	4.98
Washington	36	8,257,500	464,658	152,259	5.63
Westmoreland	53	7,230,000	353,454	82,625	4.89
Venango	-	-	-	-	-
Fulton[1]	-	-	-	-	-
[1] Broad Top Region	16	1,714,500	89,114	23,005	5.20

Bituminous Schedule Rates by Counties

Table 3 shows the bituminous schedule rates by counties. Blair and Cambria Counties have the minimum average rate, and Lawrence and Washington Counties the maximum average rate. The schedule rates for 343 mines are below the average; the rates for 363 mines are above the average.

Losses According to Severity of Injury

Tables 4 and 5 show losses according to severity of injury for anthracite and bituminous mining for the 5-year period 1926-1930. During this period there were 78 disabling accidents per fatality in anthracite and 102 disabling accidents per fatality in bituminous; 2.35 noncompensable accidents per compensable accident in anthracite as against 1.52 in bituminous; and that the average medical cost of all injuries, including fatalities, amounted to approximately one-fourth of the compensation cost. The average cost for all injuries is 20 per cent less for bituminous mining than for anthracite mining, while the number of compensable accidents per million dollars shown on the pay-roll is 21 per cent higher in bituminous than in anthracite.

In anthracite mining the average cost of a permanent total disability is about 30 per cent more than the cost of a fatality, while in bituminous mining the cost is about 42 per cent more.

The average cost of a fatality is only approximately 9 per cent more than the average cost for the loss of an arm; this seemingly small difference may be explained by the fact that in many of the fatality cases the deceased is unmarried and has no dependents, and that therefore little or no compensation is paid.

The "pure premium" shown in Tables 4 and 5 is that part of the rate which represents the provision for the payments of losses.

PAY-ROLL AND PREMIUM COST PER TON AND COMPENSATION LOSS PER TON, BITUMINOUS MINES

In Table 6 it is seen that (1) the pay-roll cost per ton of coal mined has decreased 33 per cent from 1921 to 1930; (2) the premium cost per ton has increased 12 per cent; (3) the compensation loss per ton has increased 32 per cent; and (4) the days lost per ton have increased 8 per cent from 1924 to 1930.

PRODUCTION BY COUNTIES, ALSO TONS AND PERCENTAGE WORKING ON DIFFERENT WAGE SCALES

Table 7 is self-explanatory.

ANALYSIS OF MEDICAL COST

Table 8 shows (1) the number of all accidents in anthracite mining decreased 58 per cent during the 5-year period 1926 to 1930, and in bituminous mining the decrease amounted to 39 per cent; while the medical cost in anthracite decreased 48 per cent, and in bituminous mining the decrease was 34 per cent; (2) the average medical cost per case has increased 19 per cent in anthracite mining and 8 per cent in bituminous mining during the past 5 years; (3) the decrease in the number of compensable accidents during the 5-year period was 35 and 15 per cent, respectively, for anthracite and bituminous mines. The decrease was really greater than shown by these figures, because in 1928 the waiting period was reduced from 10 days to 7 days, thereby increasing the number eligible for compensation.

Table 4.- Loss analysis by severity of injury in anthracite mining, 1926-1930, converted to 1928 level of benefits but does not include catastrophes

Severity of injury	Compensable accidents	Total cost	Compensation	Funeral and medical costs	Average cost	Average compensation	Average medical costs	Compensable accidents per million dollars on pay roll	Pure premium
1	2	3	4	5	6	7	8	9	10
All injuries	4,229	$1,673,525	$1,351,191	$322,334	$396	$320	$76	69.3	$2.74
Death	174	780,116	747,920	32,196	4,483	4,298	185	2.8	1.28
Permanent total	10	63,890	61,410	2,480	6,389	6,141	248	.2	.11
Major permanent:	123	275,245	252,190	23,055	2,238	2,050	188	2.0	.45
Loss of arm	5	19,579	18,485	1,094	3,916	3,697	219	.1	.03
Loss of hand	5	13,532	12,678	854	2,706	2,535	171	.1	.02
Loss of leg	7	24,830	23,475	1,355	3,547	3,354	193	.1	.04
Loss of foot	8	18,499	16,952	1,547	2,312	2,119	193	.1	.03
Loss of eye	30	65,314	61,460	3,884	2,178	2,049	129	.5	.11
Disfigurement	21	12,582	10,996	1,586	599	524	75	.3	.02
Other permanent	35	101,085	90,944	10,141	2,888	2,598	290	.6	.17
Indeterminate	12	19,794	17,200	2,594	1,649	1,433	216	.2	.03
Minor permanent	84	36,909	30,502	6,407	439	363	76	1.4	.06
Temporary	3,838	421,276	253,005	168,271	110	66	44	62.9	.69
Noncompensable	(1,541)	21,089	6,164	14,925	14	4	10	(25.2)	.03
	(8,000)	75,000	-	75,000	9	-	9	(131.1)	.12
(Total medical)	-	-	-	(322,334)	-	-	-	-	(.53)

Table 5.- Loss analysis by severity of injury in bituminous mining, 1926-1930, converted to 1928 level of benefits but not including catastrophes

Severity of injury	Compensable accidents	Total cost	Compensation	Funeral and medical costs	Average cost	Average compensation	Average medical costs	Compensable accidents per million dollars on pay roll	Pure premium
1	2	3	4	5	6	7	8	9	10
All injuries	33,956	$10,760,258	$8,635,063	$2,125,195	$317	$254	$63	87.9	$2.78
Death	817	3,107,706	2,957,480	150,226	3,804	3,620	184	2.1	0.80
Permanent total	157	1,031,546	991,370	40,176	6,570	6,314	256	.4	.27
Major permanent:	997	2,364,585	2,128,852	235,733	2,372	2,135	237	2.6	.61
Loss of arm	24	87,482	81,436	6,046	3,645	3,393	252	.1	.02
Loss of hand	65	177,233	165,113	12,120	2,727	2,540	187	.2	.05
Loss of leg	57	201,535	181,785	19,750	3,536	3,189	347	.2	.05
Loss of foot	84	210,629	187,247	23,382	2,507	2,229	278	.2	.05
Loss of eye	196	395,714	367,642	28,072	2,019	1,876	143	.5	.10
Disfigurement	49	30,947	24,786	6,161	632	506	126	.1	.01
Other permanent	273	790,249	707,092	83,157	2,895	2,590	305	.7	.21
Indeterminate	249	470,796	413,751	57,045	1,891	1,662	229	.6	.12
Minor permanent	616	313,018	264,123	48,895	508	429	79	1.6	.08
Temporary	31,369	3,519,551	2,265,526	1,254,025	112	72	40	81.2	.91
Noncompensable	(6,928)	86,040	27,712	58,328	12	4	8	(17.9)	.02
	(43,564)	337,812	-	337,812	8	-	8	(112.7)	.09
(Total medical)	-	-	-	2,125,195	-	-	-	-	(.55)

INSURANCE EXPERIENCE BY COUNTIES - BITUMINOUS MINING

Table 9 shows the bituminous insurance experience by counties. Based on the number of accidents per million dollars of pay-roll it is seen that Elk, Centre, and Blair Counties have the highest rates. In considering the different mining districts it is seen that the northwest district has the highest accident rate per million dollars pay-roll, with the central district and the southwestern district next in order.

INSURANCE EXPERIENCE BY BEDS OF COAL

The insurance experience by beds of coal for bituminous mining is shown in Table 10, which includes the experience of 9,514 mines over a 5-year period. It is observed that about 89 per cent of the mines operate in the Pittsburgh, Freeport, and Kittanning beds, and that the number of accidents per million dollars of pay-roll at mines operating in these three beds are approximately the same. The average compensation cost per accident is slightly higher for accidents occurring in the Freeport coals, with the Pittsburgh and the Kittanning beds next in order.

Table 6.- Pay-roll and premium cost per ton, compensation loss per ton, also number of compensable accidents per million tons at insured bituminous mines in Pennsylvania

Year	Insured tonnage	Audited pay-roll	Premium	Incurred losses	Days lost	Pay-roll cost per ton	Premium cost per ton	Compensable loss per ton	Days lost per ton
1	2	3	4	5	6	7	8	9	10
1926-1930	306,182,380	$382,809,769	$11,594,232	$9,780,761	9,327,667	$1.250	$0.0379	$0.0319	0.0305
1930	43,786,454	47,615,752	1,937,085	1,621,763	1,436,400	1.087	.0442	0.0370	0.0328
1929	60,314,824	65,277,294	2,422,197	2,114,616	1,879,048	1.082	.0402	.0351	.0312
1928	57,777,389	71,086,308	2,602,732	2,128,507	1,820.937	1.230	.0450	.0368	.0315
1927	66.062,970	90,961,037	2,465,706	1,861,047	1,969,282	1.377	.0373	.0282	.0298
1926	78,240,743	107,869,378	2,166,512	2,054,823	2,222,000	1.379	.0277	.0263	.0284
1925	71,299,417	96,644,898	1,900,192	1,939,368	2,079,250	1.355	.0267	0.0272	0.0292
1924	60,317,752	106,426,721	2,049,030	1,784,055	1,809,600	1.764	.0340	.0296	.0300
1923	87,684,545	151,196,591	3,144,783	2,259,052	-	1.724	.0359	.0258	-
1922	63,878,849	127,468,600	2,658,234	1,703,200	-	1.995	.0416	.0267	-
1921	66,196,031	115,256,600	2,584,892	1.673,250	-	1.741	.0390	.0253	-

1 - 1930 is 85 per cent complete, there having been 51,455,382 tons at insured mines.

2 - Days lost on basis of standard weights of International Industrial Accident Commission.

Table 7.- Production by counties as reported by State Department of Mines showing tons and percentage insured in 1930, also tons and percentage working on different wage scales as of July 30, 1930

County	Total coal mined, tons	Insured		Daily wage scale of insured tonnage					
		Tons	Per cent of total tons	$5.50	$5.00	$4.50	$4.00	$3.50	Tonnage idle
All counties	123,417,450	51,455,382	41.7	4,955,291	15,067,627	15,602,313	11,131,248	1,955,727	2,743,176
Allegheny	15,435,454	8,009,304	52	3,438,013	2,420,676	1,194,342	315,712	319,317	321,244
Armstrong	3,503,760	1,155,623	33	-	18,155	291,025	694,022	-	152,421
Blair	281,313	266,791	95	-	266,791	-	-	-	-
Butler	947,116	947,116	100	-	227,690	674,755	-	-	44,671
Cambria	15,274,503	9,146,908	60	-	5,629,291	3,153,291	133,545	-	230,780
Centre	694,069	485,241	70	-	7,823	-	477,418	-	-
Clarion	1,280,185	675,228	53	-	-	146,598	339,673	93,522	95,435
Clearfield	3,465,081	2,082,692	60	-	414,713	397,032	895,864	178,828	196,255
Clinton and Elk	991,607	183,965	19	-	-	-	183,965	-	-
Fayette	23,789,159	5,871,398	25	-	1,293,516	2,236,276	1,812,402	416,861	112,343
Greene	4,880,033	4,309,199	88	-	3,539,676	149,052	106,381	256,774	257,316
Indiana	7,614,616	1,876,445	25	230,693	182,450	913,737	390,483	25,589	133,493
Jefferson	2,265,585	599,951	26	-	-	-	557,380	-	42,571
Mercer-Beaver-Lawrence	508,137	181,971	36	-	98,492	44,577	38,902	.	-
Somerset	8,161,612	5,555,256	68	-	476,656	2,077,331	2,546,93[illegible]	454,336	-
Washington	19,049,869	4,963,201	26	-	235,524	1,569,989	1,851,382	208,700	1,097,600
Westmoreland	14,012,550	4,766,295	34	1,286,585	202,016	2,461,456	755,391	1,800	59,047
Broad Top	925,890	292,852	32	-	-	292,852	-	-	-
All others	335,911	85,946	26	-	54,157	-	31,789	-	-
Per cent of insured tonnage	-	-	-	9.6	29.3	30.3	21.7	3.8	5.3

1 - Includes small tonnage at $2.50 and $3.00 per day.

2 - Per cent idle — Basis

Per cent idle	Basis
0.6	$5.50
1.5	5.00
1.4	4.50
1.7	4.00
0.1	3.50

Table 8.- Analysis of medical costs by policy years

Policy year	All accidents			Compensable accidents			Noncompensable accidents		
	Number	Medical	Average per case	Number	Medical	Average per case	Number	Medical	Average per case
	ANTHRACITE MINING								
1926-1930	13,770	$318,584	$23.14	4,229	$228,659	$54.07	9,541	$89,925	$9.43
1930	1,894	47,161	24.90	728	36,462	50.09	1,166	10,699	9.18
1929	2,052	55,713	27.15	738	43,068	58.36	1,314	12,645	9.62
1928	2,426	60,103	24.77	805	45,374	56.37	1,621	14,729	9.09
1927	2,886	64,824	22.46	838	46,162	55.09	2,048	18,662	9.11
1926	4,512	90,783	20.12	1,120	57,593	51.42	3,392	33,190	9.78
	BITUMINOUS MINING								
1926-1930	84,448	$2,112,145	25.01	33,956	$1,716,005	$50.54	50,492	$396,140	$7.85
1930	11,765	312,386	26.55	5,264	262,684	49.90	6,501	49,702	7.65
1929	16,891	424,740	25.15	7,144	354,024	49.56	9,747	70,716	7.26
1928	17,349	440,751	25.40	7,297	368,155	50.45	10,052	72,596	7.22
1927	19,010	460,674	24.23	7,021	366,174	52.15	11,989	94,500	7.88
1926	19,433	473,594	24.37	7,230	364,968	50.48	12,203	108,626	8.90

Table 9.- Insurance experience by bituminous counties, exclusive of catastrophes, 1926-1930

County	Incurred losses	Compensable accidents	Accidents per million dollars of pay-roll	Average rate	Pure premium	Loss ratio
All counties	$9,943,701	33,956	88	$3.12	$2.57	0.82
1. Allegheny	1,076,258	3,033	68	2.94	2.40	0.82
2. Armstrong	324,257	1,133	102	3.27	2.93	.90
3. Blair	46,321	189	110	2.72	2.71	.99
4. Butler	362,317	1,090	96	3.47	3.18	.91
5. Cambria	1,456,688	5,621	91	2.94	2.37	.80
6. Centre	105,300	303	113	3.46	3.94	1.14
7. Clarion	178,834	627	95	3.49	2.71	.78
8. Clearfield	540,869	1,907	87	3.23	2.47	.77
9. Elk	34,442	142	115	3.59	2.80	.78
10. Fayette	887,526	3,543	87	3.13	2.19	.70
11. Greene	707,103	2,444	81	3.32	2.35	.71
12. Indiana	416,920	1,402	99	3.02	2.95	.98
13. Jefferson	231,978	768	106	3.36	3.22	.96
14. Mercer	38,725	91	94	3.73	3.99	1.07
15. Somerset	1,109,144	3,739	80	2.90	2.38	.82
16. Tioga	22,278	60	77	3.22	2.88	.89
17. Washington	1,174,195	3,535	104	3.40	3.45	1.02
18. Westmoreland	1,000,409	3,639	86	3.09	2.38	.77
19. Broad Top Region	115,054	345	85	3.66	2.83	.77
20. All others	115,083	345	109	3.55	3.62	1.02
1-10-11-17-18	4,845,491	16,194	85	3.15	2.53	.80
2-4-7-14-20	1,019,216	3,286	99	3.42	3.07	.90
3-5-6-8-9-12-13-15-16-19	4,078,994	14,476	89	3.03	2.52	.83

Table 10.- Insurance experience by bituminous beds of coal, 1926-1930

Name of bed	Incurred losses	Number of compensable accidents	Accidents per million dollars of pay-roll	Average compensation rate
All beds	$9,943,701	33,956	88	$3 12
Pittsburgh	3,527,518	12,076	88	3.27
Sewickley and Redstone	192,264	731	99	3.00
Freeport coals:	2,465,673	7,794	85	3.05
Thick Freeport	830,443	2,183	62	2.75
Upper Freeport	1,033,679	3,555	99	3.19
Lower Freeport	601,551	2,056	99	3.33
Kittanning coals:	3,545,294	12,682	88	3.01
Upper Kittanning	199,465	724	67	3.34
Middle Kittanning	627,814	1,794	101	3.15
Lower Kittanning	2,718,015	10,164	89	2.96
Clarion and Brookville	191,963	626	102	3.55
All others	20,989	47	96	3.56

Table 11.- List of catastrophes (accidents causing five or more deaths) in insured bituminous mines, 1916-1929[1]

Year	Cause of accident	Deaths	Actual cost	1928 level cost
All	-	431	$1,574,524	$1,711,772
1916	Explosion	8	22,334	28,451
1917	do.	14	47,032	60,212
1918	do.	8	21,754	28,706
1920	do.	9	19,696	24,793
1920	Car falling into shaft	6	26,442	32,923
1922	Explosion	5	16,777	20,880
1922	do.	77	241,842	302,511
1923	do.	36	139,374	174,023
1928	do.	5	19,642	19,642
1928	do.	10	24,312	24,310
1928	do.	194	725,558	725,558
1928	do.	13	66,445	66,445
1929	do.	46	203,318	203,318

1 - During 1916-1930 in insured and self-insured bituminous mines in Pennsylvania there were 6,018 fatalities recorded by the Department of Mines; 544 or 9.04 per cent of these were in catastrophes.

Table 12.- Weekly wage distribution - all compensable accidents

	1926	1927	1928	1929	1930
ANTHRACITE MINING					
Average weekly wage	$35.88	$36.61	$34.49	$35.84	$36.74
Average weekly compensation	11.95	11.98	14.90	14.96	14.90
Ratio of weekly compensation to weekly wage	.333	.327	.432	.417	.406
BITUMINOUS MINING					
Average weekly wage	$32.89	$32.67	$29.10	$28.37	$26.88
Average weekly compensation	11.95	11.92	14.55	14.48	14.24
Ratio of weekly compensation to weekly wage	.363	.365	.500	.510	.530

Compensation Cost of Catastrophes

A list of catastrophes occurring in insured bituminous mines during 1916 to 1929 is given in Table 11. The average compensation cost per death is $3,653; adding to this the average medical and burial cost of $185, the total per death would be $3,838. The average actual cost per catastrophe was $121,117. Column 5 of this table gives the 1928 "level cost"; this is the cost figured on the increased compensation basis which became effective during 1928. The average compensation cost of a death prior to 1928 was $3,283, while the cost has increased since that time to $3,878 per death in 1930.

The compensation cost in case of catastrophes is usually a minor part of the total cost, as often the damage to property, the cost of recovery, and the loss of business amounts to many times the compensation cost.

Weekly Wage Distribution - All Compensable Accidents

From Table 12 it is seen that the compensation received from injuries is far less than the wage received. The ratio of weekly compensation to weekly wage is somewhat less for anthracite than for bituminous mining, being due to the smaller average weekly wage in bituminous mining. In anthracite mining the average weekly wage has increased 2.3 per cent, while the average weekly compensation has increased 19.8 per cent during the past 5 years; and in bituminous mining the average weekly wage has decreased 18 per cent, while the average weekly compensation has increased 16 per cent.

Compensable Accidents and Amounts of Compensation Awarded - Anthracite and Bituminous - By Counties

Table 13 shows the number of fatal, permanent disability, and temporary disability cases and amounts of compensation awarded, by counties, during the period 1926 to 1930, as reported by the Workmen's Compensation Bureau of the Pennsylvania Department of Labor and Industry.

Table 13.- Compensable accidents and amounts of compensation awarded in all insured Pennsylvania mines

FATAL CASES

County	1930		1929		1928		1927		1926		Average compen-sation cost
	Cases	Amount	Cases	Amount	Cases	Amount	Cases	Amount	Cases	Amount	
Bituminous mines											
Total	337	$1,299,674	403	$1,497,304	480	$1,824,482	423	$1,344,944	435	$1,417,868	$3,551
Allegheny	44	184,559	47	183,277	34	96,584	35	106,095	52	144,129	3,371
Armstrong	11	50,351	13	42,219	11	34,056	14	46,222	10	31,243	3,459
Beaver	1	4,460	-	-	2	9,292	-	-	1	1,033	3,696
Bedford	2	5,836	2	8,788	-	-	1	909	1	2,500	3,006
Blair	-	-	1	1,650	-	-	1	4,720	1	6,050	4,140
Bradford	-	-	-	-	-	-	-	-	-	-	-
Butler	5	16,952	5	19,884	6	29,006	7	23,764	4	14,512	3,856
Cambria	43	158,876	40	147,655	36	156,806	44	137,099	46	122,670	3,460
Cameron	-	-	-	-	-	-	-	-	-	-	-
Centre	4	12,194	-	-	2	4,150	2	6,854	2	8,307	3,151
Clarion	4	15,798	1	8,118	2	7,414	3	7,934	5	14,599	3,591
Clearfield	13	46,472	11	49,152	15	60,244	20	64,536	18	50,886	3,523
Clinton	1	3,673	-	391	2	3,126	-	-	-	-	2,136
Elk	1	2,309	3	9,470	1	3,464	2	10,034	1	2,673	3,494
Fayette	48	199,830	53	178,923	47	178,216	82	257,062	66	220,223	3,494
Fulton	-	-	-	-	-	-	-	-	-	-	-
Greene	18	65,495	36	103,299	161	689,491	22	83,864	18	71,610	3,975
Huntingdon	3	10,588	2	14,667	1	5,646	1	7,018	1	-846	4,634
Indiana	23	128,360	24	86,089	25	63,514	29	89,472	71	267,547	3,692
Jefferson	6	22,214	6	31,937	6	23,783	8	42,492	10	38,301	4,409
Lawrence	-	-	-	-	3	15,836	2	9,779	1	1,300	4,486
Lycoming	-	-	-	-	-	-	-	-	-	-	-
McKean	-	-	-	-	-	-	-	-	-	-	-
Mercer	-	-	1	1,886	1	3,684	1	2,500	2	10,117	3,637
Somerset	24	72,393	22	65,975	30	136,284	38	143,194	34	98,493	3,489
Tioga	2	9,063	-	-	-	-119	-	-	1	2,545	3,830
Venango	-	-	-	-	-	-	-	-	-	-	-
Washington	44	166,947	56	197,026	46	131,252	61	164,708	48	169,548	3,253
Westmoreland	40	123,304	80	347,690	49	172,753	50	136,688	42	140,428	3,528
Anthracite mines											
Total	479	$1,789,913	493	$1,861,115	468	$1,734,066	536	$1,692,734	414	$1,330,588	$3,518
Carbon	13	41,583	7	23,335	10	39,250	16	50,410	12	47,242	3,480
Columbia	1	1,415	2	11,532	3	13,845	6	22,599	2	8,023	4,104
Dauphin	10	29,633	11	58,105	3	11,461	9	42,942	5	25,067	4,400
Lackawanna	119	506,368	104	391,593	113	431,408	148	459,811	90	252,636	3,557
Luzerne	182	698,063	220	830,998	205	777,767	207	668,287	174	569,316	3,587
Northumberland	35	147,317	35	100,822	36	143,845	39	120,468	39	138,016	3,535
Schuylkill	116	350,008	107	429,182	97	313,114	108	321,127	90	284,033	3,277
Sullivan	-	-	2	2,265	1	4,597	1	3,155	1	2,500	2,503
Susquehanna	-	-	4	10,133	-	-2,221	2	4,642	1	3,755	2,330
Wayne	3	15,526	1	3,150	-	-	-	-707	-	-	4,492

1 - Reported by Workmen's Compensation Bureau.

Table 13.- Compensable accidents and amounts of compensation awarded in all insured Pennsylvania mines - continued

PERMANENT DISABILITY CASES

County	1930		1929		1928		1927		1926		Average compen-sation
	Cases	Amount	Cases	Amount	Cases	Amount	Cases	Amount	Cases	Amount	cost
Bituminous mines											
Total	604	$735,295	483	$653,094	508	$723,971	490	$562,524	521	$651,040	$1,287
Allegheny	87	113,601	60	82,135	79	118,206	60	76,557	59	64,824	1,378
Armstrong	16	27,479	16	8,812	23	28,440	21	29,916	20	14,766	1,139
Beaver	1	316	-	-24	3	3,045	2	1,980	1	5,000	1,474
Bedford	4	11,963	3	4,680	-	-	2	2,149	3	4,130	1,910
Blair	2	622	2	274	1	2,250	-	-	2	540	527
Bradford	-	-	-	-	-	-	-	-	-	-	-
Butler	12	18,148	17	16,086	9	11,268	12	18,791	15	23,128	1,345
Cambria	54	69,046	60	87,486	63	74,595	68	91,873	79	88,318	1,270
Cameron	-	-	-	-	-	-	-	-	1	1,500	1,500
Centre	5	2,855	1	525	3	1,712	3	2,320	5	5,736	730
Clarion	6	5,763	3	-40	8	11,813	4	7,100	6	5,108	1,102
Clearfield	22	38,671	18	27,543	27	36,594	8	2,440	23	20,443	1,283
Clinton	-	-266	3	1,602	-	-	4	2,010	1	240	448
Elk	6	3,364	2	5,825	4	2,625	3	740	3	3,120	871
Fayette	65	100,188	72	91,081	82	123,044	65	79,572	64	77,944	1,356
Fulton	1	525	-	-	-	-	1	420	-	-	473
Greene	12	21,149	17	19,029	24	31,711	20	15,512	13	18,871	1,236
Huntingdon	3	990	2	525	2	2,775	-	-	7	6,240	752
Indiana	24	32,148	45	57,463	32	43,678	52	43,915	44	65,346	1,231
Jefferson	22	28,973	7	13,549	9	12,094	13	25,640	15	17,410	1,480
Lawrence	2	2,354	1	2,699	-	-13	3	3,550	3	3,420	1,334
Lycoming	-	-	-	-	-	-	-	-	1	180	180
McKean	-	-	-	-	-	-	-	-	-	-	-
Mercer	2	2,138	1	1,452	2	2,861	1	210	1	1,800	1,209
Somerset	32	43,704	30	35,014	25	40,906	35	42,821	40	56,517	1,352
Tioga	1	300	2	5,592	1	180	1	5,000	2	2,820	1,985
Venango	-	-	-	-	-	-	-	-	-	-	-
Washington	75	110,634	74	107,384	64	109,367	62	47,913	62	75,607	1,338
Westmoreland	50	100,657	47	84,402	47	66,820	50	62,095	50	68,032	1,566
Anthracite mines											
Total	525	$629,801	502	$666,061	488	$631,994	521	$588,415	369	$429,036	$1,225
Carbon	9	9,561	6	12,475	5	3,941	11	18,790	9	10,780	1,389
Columbia	9	21,811	7	4,803	2	3,369	9	12,520	5	11,390	1,684
Dauphin	5	5,903	9	16,317	17	6,387	23	25,164	8	6,305	969
Lackawanna	112	133,943	112	155,547	110	123,396	114	154,940	89	109,644	1,262
Luzerne	205	248,359	207	246,523	214	247,291	214	219,905	158	164,803	1,129
Northumberland	45	51,460	34	57,592	44	51,891	44	53,116	26	31,098	1,270
Schuylkill	139	158,614	125	172,016	95	195,419	104	102,810	68	87,816	1,350
Sullivan	-	-	2	788	1	300	1	390	1	1,500	596
Susquehanna	-	-	-	-	-	-	-	-	4	5,340	1,335
Wayne	1	150	-	-	-	-	1	780	1	360	430

1 - Reported by Workmen's Compensation Bureau.

Table 13.- Compensable accidents and amounts of compensation awarded in all insured Pennsylvania mines - concluded

TEMPORARY DISABILITY CASES

County	1930 Cases	1930 Amount	1929 Cases	1929 Amount	1928 Cases	1928 Amount	1927 Cases	1927 Amount	1926 Cases	1926 Amount	Average compensation cost
Bituminous mines											
Total	12,688	$1,158,334	13,141	$1,250,298	12,227	$1,067,266	11,458	$885,097	11,746	$889,565	$86
Allegheny	1,827	149,027	1,869	174,805	1,[illegible]38	132,684	1,279	124,252	1,199	91,824	88
Armstrong	452	46,149	421	35,101	509	49,108	454	36,570	434	32,948	88
Beaver	18	1,181	14	2,498	31	1,505	30	1,574	34	1,829	68
Bedford	61	11,722	66	9,663	60	8,000	48	4,977	57	9,553	150
Blair	50	5,481	46	3,481	33	2,153	29	2,535	30	2,334	85
Bradford	2	92	1	32	1	28	1	12	2	142	44
Butler	191	23,852	220	24,789	229	22,215	184	6,435	215	19,362	93
Cambria	1,839	140,454	1,717	155,168	1,493	128,774	1,440	94,680	1,663	125,458	79
Cameron	4	83	1	15	-	-	-	-	1	14	19
Centre	112	8,930	81	8,086	94	9,345	91	6,519	36	7,269	87
Clarion	182	15,656	160	14,098	142	8,907	131	12,749	126	11,598	85
Clearfield	484	37,338	524	50,307	433	42,970	527	34,165	507	38,271	82
Clinton	21	1,294	40	4,046	24	1,330	27	1,104	26	1,146	65
Elk	97	5,374	87	8,095	83	5,212	73	4,693	75	3,526	65
Fayette	1,509	156,486	1,862	159,735	1,756	145,211	1,741	141,686	1,657	116,225	84
Fulton	8	576	17	931	8	552	17	1,058	18	1,472	67
Greene	410	38,691	571	46,331	560	36,629	517	35,511	434	26,040	74
Huntingdon	75	6,595	64	4,424	37	2,017	51	2,277	68	6,209	73
Indiana	738	72,122	705	89,305	824	89,322	735	51,750	862	69,976	96
Jefferson	262	21,816	316	39,071	255	22,004	226	16,319	294	23,340	91
Lawrence	21	1,543	37	1,850	24	1,686	28	1,149	25	2,057	61
Lycoming	8	211	-	-	-	-	3	1,056	10	490	84
McKean	-	-	-	-	-	-	2	14	1	190	68
Mercer	21	970	29	1,884	41	2,031	40	6,498	31	1,844	82
Somerset	955	80,750	919	80,379	1,049	89,663	902	63,127	832	55,684	79
Tioga	44	3,143	34	2,338	25	4,933	21	1,642	23	1,428	92
Venango	8	245	1	39	3	145	5	172	3	138	37
Washington	1,791	174,171	1,752	172,856	1,446	127,906	1,330	120,004	1,537	122,942	91
Westmoreland	1,506	154,382	1,587	160,971	1,529	132,936	1,526	112,569	1,498	113,256	88
Anthracite mines											
Total	14,484	$1,099,571	14,700	$1,097,486	13,471	$972,323	12,341	$775,285	8,972	$590,407	$71
Carbon	328	25,048	386	32,457	323	24,114	381	20,613	276	18,062	71
Columbia	173	10,585	132	11,751	116	9,361	140	10,774	136	10,872	77
Dauphin	281	19,860	283	22,306	219	16,160	164	9,404	109	9,106	73
Lackawanna	3,313	214,325	3,409	214,686	2,840	205,966	2,584	170,318	1,949	122,844	66
Luzerne	5,341	396,857	5,429	400,145	4,868	318,367	4,566	265,705	3,134	191,924	67
Northumberland	1,399	123,917	1,251	105,669	1,344	112,176	1,216	87,802	949	83,592	83
Schuylkill	3,588	304,844	3,741	306,919	3,696	281,266	3,206	207,248	2,376	152,241	75
Sullivan	24	3,072	36	2,261	26	2,543	33	1,379	18	1,014	75
Susquehanna	14	544	12	615	30	1,896	36	1,659	17	444	47
Wayne	23	519	21	677	9	474	15	383	8	308	31

1 - Reported by Workmen's Compensation Bureau.

It is shown that (1) the average compensation cost of fatal cases in bituminous mines was $3,551 and $3,518 for anthracite mines; (2) the average compensation cost of permanent disability cases for bituminous mines was $1,287 and $1,225 for anthracite mines; and (3) the average compensation cost of temporary disability cases for bituminous mines was $86 and $71 for anthracite mines. The compensation cost in all cases was slightly lower for anthracite than for bituminous mines, the greater difference being in cases of temporary disability.

There has been a substantial decrease in the number of fatal cases in bituminous mines during the 5 years, but in anthracite mining little or no decrease in the number of fatal cases has been effected.

No decrease in the number of permanent disability cases has been shown in bituminous mining during the past five years, while there has been a decided increase in these cases in anthracite mining.

There has been a general increase in temporary disability cases in both bituminous and anthracite mining during the 5-year period. The large increase in these cases in 1928 over 1927 may be accounted for by the fact that in 1928 the law changing the waiting period from 10 to 7 days became effective.

Compensable Accidents by Injury

In reviewing the compensable accidents by place of injury to the person for the 5-year period, as shown in Table 14, it will be observed that little progress was made in preventing loss of body members by accidents, and in many cases the losses have increased from year to year, with like increases in the compensation cost.

It is observed that eye-loss accounted for the greatest number of days lost, there being an average of 392,560 days lost per year from this cause in Pennsylvania mines. This time loss does not take into account that due to eye injuries where there is no loss of eyes; this amounts to an average of 1,076 man-years per year, which would be equivalent to the total employment of approximately four average Pennsylvania mines.

Despite the fact that goggles have been introduced at a number of mines where their wearing is compulsory, the decrease in eye-injury cases has been small. Some individual companies have materially reduced their eye accidents by making the use of goggles compulsory when performing certain types of work, but the industry as a whole has made little progress in reducing this type of accident.

During the 5-year period covered by Table 14 there was an average yearly loss of 200 eyes, 20 arms, 56 hands, 335 fingers, 113 parts of fingers, 57 legs, 72 feet, 100 facial disfigurements, 44 miscellaneous permanent total disabilities, and 7 miscellaneous permanent partial disabilities, or an average total of 1,102 permanent total disability injuries per year, with an average compensation cost of $1,861 per case.

Accidents According to Cause

From Table 15 it is observed that the major causes of mine accidents are: (1) Falling objects (including roof and coal falls); (2) transportation; (3) handling objects by hand; (4) hand tools; and (5) falls of persons. A summary follows Table 15.

Table 14.- Compensable accidents by place of injury in all insured mines of Pennsylvania[1]

ANTHRACITE MINES

Member lost	Year	Cases	Losses	Days lost[2]	Average days lost	Compensation	Average compensation
	1930	98	107	212,800	2,171	$211,704	$2,160
	1929	104	115	221,400	2,129	221,309	2,128
Eye	1928	92	103	204,600	2,224	179,302	1,949
	1927	126	134	256,200	2,033	208,514	1,655
	1926	97	102	195,600	2,016	160,389	1,653
	1930	12	12	54,00	4,500	37,049	3,087
	1929	13	13	57,000	4,385	36,332	2,795
Arm	1928	6	6	25,550	4,258	15,876	2,646
	1927	8	8	36,000	4,500	20,446	2,556
	1926	8	8	36,000	4,500	20,324	2,541
	1930	24	26	78,600	3,275	66,509	2,771
	1929	24	25	78,000	3,250	60,866	2,536
Hand	1928	22	28	81,000	3,682	61,119	2,778
	1927	22	23	70,200	3,191	47,997	2,182
	1926	16	17	51,600	3,225	36,376	2,274
	1930	114	125	40,950	359	50,660	444
	1929	128	158	57,300	448	69,964	547
Finger	1928	108	136	48,900	453	55,497	514
	1927	152	193	65,400	430	61,566	405
	1926	103	124	44,550	433	44,308	430
	1930	98	100	34,200	349	22,857	233
	1929	79	88	32,400	410	21,654	274
Phalanx	1928	93	100	36,700	395	23,623	254
	1927	74	83	30,000	405	15,453	209
	1926	61	68	25,200	413	13,546	222
	1930	11	17	11,750	1,068	8,402	764
	1929	12	16	11,850	988	8,978	748
Finger and phalanx	1928	8	11	7,650	956	5,767	721
	1927	10	16	12,300	1,230	7,230	723
	1926	6	8	6,600	1,100	3,840	640
	1930	15	16	70,425	4,695	44,088	2,939
	1929	19	21	90,000	4,737	60,218	3,169
Leg	1928	23	24	106,500	4,630	68,017	2,957
	1927	32	35	148,500	4,641	87,820	2,744
	1926	18	19	84,000	4,667	47,238	2,624
	1930	16	16	40,800	2,550	33,843	2,115
	1929	20	21	57,000	2,850	43,812	2,191
Foot	1928	27	28	72,600	2,689	56,140	2,079
	1927	15	15	40,200	2,680	26,577	1,772
	1926	18	21	57,000	3,167	36,800	2,044

1 - Reported by Workmen's Compensation Bureau.

2 - Days lost is based on standard weights for lost-time accidents.

Table 14.- Compensable accidents by place of injury in all insured mines of Pennsylvania - Continued[1]

Member lost	Year	Cases	Losses	Days lost[2]	Average days lost	Compensation	Average compensation
ANTHRACITE MINES							
Facial disfigurement	1930	113	-	22,365	198	$44,577	$394
	1929	80	-	16,341	204	32,132	402
	1928	89	-	18,780	211	33,573	377
	1927	59	-	15,536	263	26,420	448
	1926	29	-	10,239	353	17,283	596
Miscellaneous permanent total disability	1930	16	-	100,200	6,263	86,562	5,410
	1929	24	-	144,000	6,000	115,668	4,820
	1928	19	-	115,500	6,079	86,780	4,567
	1927	21	-	133,500	6,357	82,274	3,918
	1926	13	-	78,000	6,000	48,932	3,764
Miscellaneous permanent partial disability	1930	8	-	22,200	2,775	23,550	2,944
BITUMINOUS MINES							
Eye	1930	78	84	165,600	2,123	$148,328	$1,902
	1929	77	84	168,000	2,182	154,161	2,002
	1928	80	92	180,600	2,258	157,203	1,965
	1927	82	89	170,400	2,078	133,527	1,628
	1926	95	102	189,600	1,996	153,353	1,614
Arm	1930	15	15	66,375	4,425	44,209	2,947
	1929	6	6	27,000	4,500	17,201	2,867
	1928	16	16	72,000	4,500	44,707	2,794
	1927	8	8	36,000	4,500	20,443	2,555
	1926	10	10	45,000	4,500	25,141	2,514
Hand	1930	24	24	74,250	3,094	56,833	2,368
	1929	28	30	94,200	3,364	74,506	2,661
	1928	41	44	135,600	3,307	99,876	2,436
	1927	34	35	107,400	3,159	73,555	2,163
	1926	26	28	82,800	3,185	56,287	2,165
Finger	1930	141	171	61,200	434	70,238	498
	1929	161	205	75,450	469	84,385	524
	1928	130	157	56,700	436	63,267	487
	1927	138	169	58,500	424	56,197	407
	1926	128	163	67,650	528	58,093	454
Phalanx	1930	89	94	32,850	369	21,548	242
	1929	94	100	34,950	372	23,141	246
	1928	93	105	38,700	416	24,952	268
	1927	107	122	43,700	408	22,240	208
	1926	105	117	42,300	403	22,259	212
Finger and phalanx	1930	12	19	13,800	1,150	10,343	862
	1929	9	11	9,000	1,000	6,199	689
	1928	15	18	17,250	1,150	13,582	905
	1927	10	13	10,350	1,035	5,998	600
	1926	10	21	12,350	1,235	8,730	873

1 - Reported by Workmen's Compensation Bureau.

2 - Days lost is based on standard weights for lost-time accidents.

Table 14.- Compensable accidents by place of injury in all insured mines of Pennsylvania - Concluded[1]

BITUMINOUS MINES

Member lost	Year	Cases	Losses	Days lost[2]	Average days lost	Compensation	Average compensation
	1930	23	25	104,700	4,552	$73,009	$3,174
	1929	18	18	81,000	4,500	50,054	2,781
Leg	1928	22	23	100,500	4,568	63,037	2,865
	1927	24	24	108,000	4,500	61,596	2,567
	1926	30	31	136,500	4,550	77,786	2,593
	1930	47	48	120,240	2,558	100,577	2,140
	1929	45	46	120,600	2,680	95,301	2,118
Foot	1928	57	61	161,400	2,832	122,676	2,152
	1927	47	51	133,800	2,847	92,632	1,971
	1926	53	55	143,400	2,706	101,343	1,912
	1930	24	-	5,774	241	11,582	483
	1929	17	-	5,369	316	8,499	500
Facial disfigurement	1928	28	-	8,886	317	16,801	600
	1927	20	-	6,398	320	10,874	544
	1926	34	-	14,704	432	24,821	730
	1930	23	-	139,500	6,065	115,927	5,040
	1929	26	-	156,000	6,000	128,469	4,941
Miscellaneous permanent	1928	26	-	156,000	6,000	117,930	4,536
total disability	1927	22	-	132,000	6,000	89,590	4,072
	1926	30	-	180,000	6,000	123,234	4,108
Miscellaneous permanent partial disability	1930	28	-	74,304	2,654	79,172	2,828

1 - Reported by Workmen's Compensation Bureau.

2 - Days lost is based on standard weights for lost-time accidents.

Table 15. Accidents reported to the Workmen's Compensation Bureau, according to cause in all insured mines in Pennsylvania

BITUMINOUS MINES

Location	Total F	Total NF	Working machinery F	Working machinery NF	Boiler and pressure apparatus F	Boiler and pressure apparatus NF	Pumps and prime movers F	Pumps and prime movers NF	Transmission apparatus F	Transmission apparatus NF	Elevators and hoists F	Elevators and hoists NF	Transportation F	Transportation NF	Handling objects by hand F	Handling objects by hand NF	Hand tools F	Hand tools NF	Electricity F	Electricity NF	Explosives and explosions F	Explosives and explosions NF	Hot and corrosive substances F	Hot and corrosive substances NF	Falling objects F	Falling objects NF	Falls of persons F	Falls of persons NF	Stepping upon or striking against objects F	Stepping upon or striking against objects NF	Miscellaneous F	Miscellaneous NF
1930																																
Total	337	19,781	16	814	-	5	-	32	-	5	2	124	79	4,142	3	3,032	-	2,199	8	276	13	185	1	107	203	6,056	3	1,478	1	698	8	628
Surface	22	1,760	1	69	-	3	-	12	-	3	-	20	13	432	-	354	-	162	-	54	-	15	1	32	2	160	2	263	-	65	3	116
Shaft	3	171	-	3	-	-	-	-	-	-	2	3	1	56	-	28	-	15	-	3	-	1	-	2	-	28	-	18	-	7		7
Underground	312	17,850	15	742	-	2	-	20	-	2	-	101	65	3,654	3	2,650	-	2,022	8	219	13	169	-	73	201	5,868	1	1,197	1	626	5	505
1929																																
Total	408	23,131	8	881	-	5	1	34	-	13	6	148	89	4,940	4	3,430	-	2,615	8	271	61	238	1	148	224	7,129	6	1,584	-	881	-	814
Surface	23	2,256	1	83	-	3	-	14	-	9	1	30	14	595	1	400	-	216	1	37	2	20	1	48	-	216	2	353	-	112	-	120
Shaft	8	195	-	1	-	-	-	1	-	1	-	5	-	71	-	26	-	13	-	6	7	2	-	1	1	30	-	20	-	7	-	11
Underground	377	20,680	7	797	-	2	1	19	-	3	5	113	75	4,274	3	3,004	-	2,386	7	228	52	216	-	99	223	6,883	4	1,211	-	762	-	683
1928																																
Total	543	22,580	11	1,061	-	6	-	37	-	24	2	118	82	5,165	2	3,090	2	2,566	14	302	235	231	-	170	185	6,883	3	1,376	1	767	6	784
Surface	22	2,309	1	109	-	3	-	9	-	12	1	21	14	690	-	362	1	246	2	48	-	20	-	47	2	215	1	288	-	89	-	150
Shaft	3	142	-	3	-	-	-	-	-	-	1	3	1	65	-	13	-	6	-	5	-	-	-	3	1	17	-	14	-	6	-	7
Underground	518	20,129	10	949	-	3	-	8	-	12	-	94	67	4,410	2	2,715	1	2,314	12	249	235	211	-	120	182	6,651	2	1,074	1	672	6	627
1927																																
Total	423	23,267	7	1,051	-	4	-	48	1	108	1	42	76	5,931	5	2,886	1	2,664	20	290	26	262	-	186	243	6,899	4	1,320	1	858	38	718
Surface	38	2,629	-	91	-	2	-	15	1	21	-	7	14	876	1	405	1	266	4	41	-	23	-	61	2	190	3	367	-	110	12	154
Shaft	3	150	-	1	-	-	-	1	-	-	-	10	-	65	-	19	-	2	1	3	-	-	-	-	2	24	-	16	-	3	-	6
Underground	382	20,488	7	959	-	2	-	32	-	87	1	25	62	4,990	4	2,462	-	2,396	15	246	26	239	-	125	239	6,685	1	937	1	745	26	558
1926																																
Total	443	24,234	5	1,114	-	3	-	55	1	8	3	29	86	7,351	5	2,684	2	2,809	11	265	74	271	2	192	245	7,282	4	894	1	696	4	581
Surface	33	2,598	1	124	-	3	-	17	1	5	-	4	16	1,141	2	340	1	216	3	42	-	13	1	45	1	144	4	260	-	101	3	143
Shaft	-	144	-	-	-	-	-	1	-	1	-	16	-	64	-	12	-	5	-	4	-	-	-	1	-	9	-	16	-	9	-	6
Underground	410	21,492	4	990	-	-	-	37	-	2	3	9	70	6,146	3	2,332	1	2,588	8	219	74	258	1	146	244	7,129		618	1	586	1	432

Table 15. Accidents reported to the Workmen's Compensation Bureau, according to cause in all insured mines in Pennsylvania--Continued

ANTHRACITE MINES

Location	Total		CAUSE																											
			Working machinery		Boiler and pressure apparatus		Pumps and prime movers		Transmission apparatus		Elevators and hoists		Transportation		Handling objects by hand		Hand tools		Electricity		Explosives and explosions		Hot and corrosive substances		Falling objects		Falls of persons		Stepping upon or striking against objects		Miscellaneous	
	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF	F	NF
1930																																
Total	479	26,036	2	444	2	32	1	31	5	21	2	176	52	3,180	8	5,020	2	2,916	16	174	95	643	1	172	263	7,509	13	2,374	4	1,570	13	1,274
Surface	42	3,352	2	116	1	13		11	3	12		19	13	466		727	1	406	8	43	1	13	1	100	3	380	5	592	-	238	4	216
Shaft	7	947	-	1	-	3	-	2	-	-	-	21	1	191	1	162	-	65	-	9	2	10	-	4	2	281	1	102	-	47	-	49
Underground	430	21,737	-	327	1	16	1	18	2	9	2	136	38	2,523	7	4,131	1	2,445	8	122	92	620	-	68	258	6,843	7	2,180	4	1,285	9	1,009
1929																																
Total	518	27,543	5	412	2	32	3	33	1	32	9	146	85	3,526	13	5,225	2	3,109	12	195	78	693	-	205	279	8,041	19	2,244	1	1,784	9	0 66
Surface	40	3,615	5	107	1	9	1	14	-	19	-	25	16	546	2	776	-	406	4	40	-	16	-	115	2	428	6	592	-	273	3	249
Shaft	16	688	-	5	1	3	1	1	-	-	6	11	2	128	-	110	1	41	1	8	1	14	-	5	1	209	2	74	-	45	-	34
Underground	462	23,240	-	300	-	20	1	18	1	13	3	110	67	2,852	11	4,339	1	2,662	7	147	77	663	-	85	276	7,404	11	2,178	1	1,466	6	983
1928																																
Total	473	25,417	4	442	-	30	2	37	1	30	10	134	76	3,466	8	4,940	1	2,860	6	177	79	636	-	170	260	7,230	17	2,555	3	1,664	6	1,046
Surface	35	3,463	3	131	-	8	1	16	1	14	1	26	15	538	2	785	1	407	-	33	1	17	-	83	3	377	5	541	1	283	1	204
Shaft	10	579	-	5		2	-	-	-	-	5	25	1	125	-	100	-	34	-	7	1	9	-	3	-	157	3	56	-	29	-	27
Underground	428	21,375	1	306	-	20	1	21	-	16	4	83	60	2,803	6	4,055	-	2,419	6	137	77	610	-	84	257	6,696	9	1,959	2	1,352	5	215
1927																																
Total	536	26,817	3	376	-	30	-	30	5	111	4	76	59	4,522	11	5,081	2	2,897	7	153	112	770	-	179	265	6,820	15	2,628	2	1,959	50	1,185
Surface	54	4,286	2	135	-	7	-	16	5	32	-	9	12	780	2	931	-	478	3	32	2	27	-	123	11	391	3	710	-	364	14	253
Shaft	12	713	-	6	-	2	-	-	-	2	2	33	1	188	-	106	-	46	-	5	4	13	-	3	3	154	1	75	-	45	1	35
Underground	470	21,818	1	235	-	21	-	14	-	77	2	34	46	3,554	9	4,044	2	2,373	4	116	106	730	-	53	251	6,277	11	1,843	2	1,550	35	897
1926																																
Total	484	27,633	2	546	3	9	-	59	1	10	6	49	76	6,317	6	4,772	2	3,073	6	167	102	673	11	258	246	6,907	12	2,030	2	1,714	9	1,049
Surface	36	4,865	2	264	3	3	-	28	-	4	2	7	15	1,179	2	984	-	559	3	34	1	33	1	117	4	332	2	560	-	451	1	260
Shaft	5	672	-	5	-	-	-	3	-	1	2	18	1	184	-	104	-	51	-	6	-	15	-	7	2	166	-	45	-	40	-	27
Underground	443	22,096	-	277	-	6	-	28	1	5	2	24	60	4,954	4	3,684	2	2,463	3	127	101	625	10	134	240	6,359	10	1,425	2	1,223	8	762

1 - Includes accidents causing two or more days lost time as reported by Workmen's Compensation Bureau.

Summary of Major Causes of Accidents
as given in table 15, per cent

Cause	Anthracite		Bituminous	
	Fatal	Nonfatal	Fatal	Nonfatal
Falling objects	52.8	30.8	51.3	30.3
Transportation	14.0	17.7	19.4	24.4
Handling objects	1.8	21.1	0.88	13.4
Hand tools	0.36	12.5	0.23	11.4
Falling persons	3.0	11.0	0.93	5.88

It is observed that Table 15 records the number of accidents causing two or more days lost time reported to the Workmen's Compensation Bureau; this is about 27 per cent more than the cases awarded compensation.

Cost of Accidents

That little progress has been made in reducing accidents in Pennsylvania coal mines during recent years is shown in Table 16. However, the number of fatalities in both anthracite and bituminous mines in the State during 1931 was lower than for any previous year. Some individual companies who have made sincere and intelligent efforts toward preventing accidents have made remarkable records, resulting materially in a reduction in the cost of production. One bituminous coal mining company which operates 13 mines and produces approximately two million tons of coal annually, through organized efforts in promoting safety has made a saving of approximately $60,000 in compensation cost in one year. This is a saving of 3 cents a ton, which, added to the medical cost and other accident costs, may mean the difference between profit and loss during the year.

The savings effected in the reduction of accidents are greater by far than is at first apparent. Generally, when accident cost savings are figured, compensation and medical cost only are taken into consideration because this is a cost which may be found readily, even though many companies do not obtain or use it. Other factors that enter into the cost of accidents and which can not be readily calculated in money values are conservatively estimated to be 4 times the compensation cost. In other words, a reduction of 1 cent per ton in compensation cost would actually be a reduction of 5 cents per ton in the cost of producing coal.

From Table 16 it is observed that the average production per compensable accident in bituminous mining in Pennsylvania was 10,408 tons, whereas the production per accident in 1930 was 9,199 tons. In anthracite mining the average production per compensable accident was 5,463 tons, and the production per accident for 1930 was 3,999 tons.

The compensation and medical cost for bituminous mining in Pennsylvania for 1930 was 3.22 cents per ton, and the average for the five years was 2.93 cents per ton.[3] For anthracite mining this cost for 1930 was 7.26 cents per ton, with an average of 5.43 cents per ton. The present average cost per ton would be considerably more than shown in the table for the 5-year period, inasmuch as in 1928 the amount of compensation paid to injured employees was

3 - In a paper presented at the Twenty-Second Annual Meeting of the Mine Inspector's Institute of America, Richmond, Va., May 4, 5, and 6, 1931, Rush N. Hosler, superintendent, Coal Mine Section, Pennsylvania Compensation Rating and Inspection Bureau, estimated the compensation and medical cost of accidents in Pennsylvania bituminous mines to be 3.6 cents per ton, or a total of more than $5,000,000 a year.

Table 16.- Compensation and medical cost of accidents

Year	Coal produced, tons	Compensable accidents	Compensation cost	Medical cost[1]	Compensation + medical cost	Compensation + medical cost per ton, cents
BITUMINOUS MINES						
1931	97,276,000	-	-	-	-	-
1930	124,462,787	13,529	$3,193,303	$798,326	$3,991,629	3.22
1929	143,516,241	14,027	3,400,696	850,174	4,250,870	2.96
1928	131,202,163	13,215	3,615,719	903,927	4,519,648	3.44
1927	133,141,639	12,371	2,792,565	698,141	3,490,706	2.62
1926	153,041,638	12,702	2,958,732	764,683	3,723,415	2.43
Avg.	137,072,894	13,169	3,192,203	803,451	3,995,654	2.93
ANTHRACITE MINES						
1931	59,531,000	-	-	-	-	-
1930	61,950,747	15,488	$3,519,285	$879,821	$4,399,106	7.26
1929	73,828,115	15,695	3,624,662	906,165	4,530,827	6.14
1928	75,348,069	14,427	3,338,383	834,596	4,172,979	5.55
1927	80,095,564	13,398	3,056,434	764,108	3,820,542	4.77
1926	84,437,452	9,755	2,350,031	587,508	2,937,539	3.43
Avg.	75,131,989	13,753	3,175,559	794,439	3,969,998	5.43

1 - Calculated.

increased and the waiting period was decreased. This fact would also partly explain the large increase in the number of accidents and amount of compensation in 1928, as compared with 1927. Using the ratio of 4 to 1, it is seen that the total accident cost for bituminous mining during 1930 was around or over 16 cents per ton, and around or over 36 cents per ton for anthracite mining.

An important factor not generally given proper recognition when considering the effect of accident costs is that when a substantial reduction is made in accident cost the operator may be able to find greater market for his product in competition with mines where accident costs are high.

During 1930, 1,336 bituminous mines in Pennsylvania produced 10,000 tons of coal or more; it is estimated that the cost of accidents during the year averaged $14,938 per mine.

The cost of accidents is ultimately passed on to the consumer and to the general public and it is estimated that the cost of coal-mining accidents in Pennsylvania during 1930 amounted to about $4.36 per capita, based on a population of 9,631,350. Moreover, one authority[4] declares that the industrial accident cost represents a tax of 11 per cent annually on the income of salaried workers.

CONCLUSIONS

1. A study of the frequency and severity of Pennsylvania coal-mining accidents during the years 1926 to 1930 shows that while there had been some decrease in fatalities in the

4 - Heinrich, H. W., Cost of Industrial Accidents to the State, the Employer, and the Man: Page 8, Reprint of address before 17th annual meeting, Internat. Assoc. Ind. Accident Boards and Commissions, Wilmington, Del., Sept. 22-26, 1930.

bituminous mines there has been an increase in nonfatal accidents, and in anthracite mines there has been an increase in both fatal and nonfatal accidents.

2. Due largely to decreased production, with its decreased payrolls and an increase in the number of accidents, the compensation cost has steadily increased during the past 5 years. In bituminous mining the increased accident cost is chiefly a result of increase in nonfatal accidents.

3. Unless there is an incentive other than that based wholly on humanitarian principles, many employers are not likely to give the subject of accident prevention due consideration as compared with production and marketing; and not until such time as the mine management fully appreciates the importance of the relationship of accident prevention to the cost of production may any material reduction in accidents be expected. It is believed that the average mine could effect at least 50 per cent reduction in accident frequency and cost with little or no expenditure of money; this can be done, however, only by sincere and concerted efforts on the part of the management, coupled with a sincere effort to enlist the cooperation of the workers.

4. During the past few years safety in mining has been given considerable impetus by Federal, State, and local organizations, and by individual companies, and many excellent safety performances have been the result. This indicates that coal mining is not so inherently hazardous that it can not be done with relative safety. With a continuance of effort on the part of those already engaged in accident-prevention work and with more widespread appreciation of the immense waste due to occurrence of accidents in mines, a general reduction in accidents in mining may be expected. There is no good reason why the excellent coal-mine fatality rate established in 1931 (the best in the present century) should not be much better in 1932 and in every year thereafter, not only in Pennsylvania but in every mining State of the Union.

I. C. 6619 May, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE BRITANNIA MINING AND SMELTING CO., LTD., BRITANNIA BEACH, B. C.

BY

A. C. MUNRO AND H. A. PEARSE

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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE BRITANNIA MINING AND SMELTING CO., LTD., BRITANNIA BEACH, B. C.[1]

By A. C. Munro[2] and H. A. Pearse[3]

INTRODUCTION

This paper, which describes the milling practice at the Britannia concentrator, Britannia Beach, B. C., is one of a series being prepared by the Bureau of Mines.

The concentrator treats copper ore at the present rate of 6,000 to 7,000 tons of dry ore daily and produces copper concentrate and pyrite concentrate by selective flotation methods. The copper concentrate is smelted and the pyrite product is utilized for acid manufacture.

ACKNOWLEDGMENTS

The authors acknowledge the courtesy of the management of the Britannia Mining and Smelting Co., Ltd., in permitting the publication of this paper. Much credit is due the department heads, mill foremen, and operators of the Britannia concentrator for their skillful contributions toward the attainment of the results herein recorded. O. Wiser, consulting metallurgist of the Howe Sound Co., gave the authors valuable assistance and co-operation.

LOCATION

The concentrator is on tidewater at Britannia Beach, Howe Sound, B. C. It is about 30 miles distant from the city of Vancouver, from which it is reached by daily steamer in two and a half hours. The mine portal and townsite are 2 miles distant from the beach at an elevation of 2,100 feet above sea level.

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1 - The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6619."

2 - One of the consulting engineers, U. S. Bureau of Mines, and general superintendent of mills, Britannia Mining and Smelting Co. (Ltd.).

3 - One of the consulting engineers, U. S. Bureau of Mines, and metallurgist, Britannia Mining and Smelting Co. (Ltd.).

METHODS OF MINING

Most of the ore is obtained by shrinkage stoping, although glory-hole mining with drawing through bulldozing chambers and square-set stoping methods are also employed. The ore is delivered to the concentrator through a system of underground raises and haulage levels. Electric locomotives, drawing trains of ten 17 or 20 ton capacity cars, are used on the main haulage lines.

POWER

The company operates a 6,000-kilowatt capacity hydroelectric plant at Britannia Beach. This plant supplies nearly all power requirements for mining and milling during flush water seasons. At other times auxiliary power is transmitted over a high-tension line from Vancouver.

WATER

The elevation of the power plant is slightly higher than that of the concentrator, so that part of the discharge from the impulse wheels of the former is utilized as the fresh water supply for milling purposes. The water contains small amounts of calcium and other salts and is usually slightly acid, the pH value varying from 6.0 to 7.0 depending upon climatic conditions. The impurities have no observed harmful effects on milling operations, excepting that variation in the acid content causes small variations in the amount of lime used in the flotation plant.

ORE TREATED

The Britannia mines comprise five individual mines spaced at intervals for a total distance of 7,000 feet along a wide and steeply dipping shear zone.[4,5] The orebodies occur as sulphide replacements in chlorite schists; the latter are derived from the shearing of quartz diorite porphyry sills. The typical schist, which is mottled with large masses of chlorite, is altered in places to a mass of tough quartz which breaks with a conchoidal fracture. The rock forming the footwall of the orebodies, and generally the hanging wall as well, is slate into which tongues or "wedges" of the porphry have been intruded.

The chief copper mineral of the ore is chalcopyrite; minor amounts of chalcocite and bornite are also present; pyrite occurs in considerable quantities, varying in different sections of the mine; minor amounts of gold and silver are also contained. Certain parts of the mine produce moderate amounts of zinc blende and also traces of galena; isolated occurrences of gypsum, calcite, magnesite, and barite have been noted.

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4 - Schofield, S. J., The Britannia Mine, British Columbia: Econ. Geol., vol. 21, May, 1926, pp. 271-284.

5 - Moore, J. I., Operations at the Britannia Mines: Eng. and Min. Jour., vol. 122, Dec. 11, 1926, pp. 924-930.

The sulphide minerals are separated fairly completely from the gangue material at 48 mesh, but grinding to 200 mesh is necessary to free the chalcopyrite from the pyrite completely.

The ore from the higher levels of the mine, from which about two-thirds of the present mill feed is being extracted by shrinkage stoping and caving, is highly oxidized and contains large quantities of primary slimes and soluble salts. These slimes, minus 65-mesh material, amount to about 13 per cent of the original mill feed and are deep yellow in color, due to the presence of iron oxide. The ore from the lower levels is clean and unoxidized but becomes contaminated with the soluble salts and slimes from the upper levels while passing through the main raises enroute to the concentrator.

Britannia ores are very hard and therefore show unusual resistance to crushing and grinding. Recent tests by an independent company indicate the "grindability" of a composite sample to be 1.57, as compared with 1.44 for Portland ore; the latter is used as a standard on account of its extreme hardness. On the same scale, Southwestern porphyry ores are about 2.5 and a dense sulphide ore about 3.2.

The moisture content of the concentrator heads varies greatly from a minimum of 3 per cent during dry seasons to a maximum of 10 per cent during spring thaws and after periods of heavy rainfall.

HISTORY OF CONCENTRATOR OPERATIONS

The first mill for the treatment of Britannia ores, known as the No. 1 concentrator, was erected in 1904 and had a capacity of 200 tons of ore per day. The capacity was increased gradually from 200 tons in 1904 to 600 tons per day in 1916. It is interesting to note that the No. 1 concentrator contained one of the first flotation units to be operated on this continent--a 600-ton capacity Minerals Separation machine installed in 1912.

In 1916 the No. 1 plant was replaced by the No. 2 concentrator, which reached a daily capacity of 2,000 tons of ore in 1921. A fire destroyed the plant in the same year. The No. 2 plant used jigging and bulk flotation methods of treatment. The combined jig and flotation concentrates produced contained about 11 per cent of copper; the pyrite was not depressed.

The present, or No. 3 concentrator, completed early in 1923, was designed to have a capacity of 2,500 tons of ore daily. According to the original plan, primary crushing to minus 3-inch ring size was done at the mine. The intermediate crushing equipment at the concentrator comprised one set of 72-inch and four sets of 54-inch rolls which operated in circuit with Hum-mer screens. These units were intended to handle only moderately dry and nonslimy ore. The condition of the ore received at the concentrator was such, however, that rolls choked and screens were blinded continuously, so that the installation of washing screens ahead of the rolls became immediately necessary. The original fine-grinding equipment consisted of eighteen 7 by 10 foot Traylor mills, each designed to be operated by a 75-hp. motor

and with a pebble charge. Six 14-cell Minerals Separation machines were provided for flotation; no provision was made for the cleaning of flotation products except that of returning middling froths of the tail end cells to the heads of the machines. The following tabulation gives, in chronological order, the changes adopted in the No. 3 concentrator since 1923.

Record of changes made in No. 3 concentrator since 1923

Date	Change
1923, March	No. 3 concentrator started operating; washing screens were installed ahead of rolls to remove fines and slimes.
1923, August	Flotation circuit was changed from neutral to alkaline for depression of pyrite.
1923, September	One additional 14-cell Minerals Separation machine was installed for cleaning rougher concentrates.
1924	Removal of primary slimes from the undersize of the washing screens by classifiers and the separate flotation treatment of this material. The addition of sulphurous acid to the cleaner flotation machine to correct the over-oiled condition of the froth; this practice increased the grade of concentrates but had a slightly injurious effect on recovery.
1925	The production of copper-iron middlings from the last five cells of the roughers; regrinding and retreatment of these middlings resulted in the production of waste iron tailings and a higher copper recovery. Experimental work with xanthate reagent was adopted on full mill scale.
1926	Xanthate reagent replaced coal tar and creosote mixtures formerly used. The addition of sulphurous acid to the cleaner machine was discontinued.
1926, November	Bulk flotation method was adopted in the rougher circuit; ball mill and classifiers were installed to regrind rougher concentrates.
1926	Gyratory crushers were transferred from the mine to the mill.
1927	The mill started to produce pyrite concentrates; Forrester flotation machines were installed for this operation. Four additional ball mills were provided for fine grinding.

Record of changes made in No. 3 concentrator since 1923—Continued

Date	Change
1928	Forrester machines added for scavenger flotation of rougher tailings. An additional mill for the regrinding of concentrates. Hydraulic classifier and grinding mill installed for the separation and regrinding of oversize material from the tailings. The addition of a small quantity of aerofloat No. 25 reagent to the roughing circuit.
1929	Elevator and baffled launder system installed in crushing plant to secure more efficient washing and separating of slimes from the ore, thus making finer screening possible. A 50-foot Forrester machine was installed to provide additional roughing capacity.
1930	Two 5 1/2-foot Symons cone crushers replaced the gyratory crushers; the 72-inch rolls were placed in parallel with the 54-inch rolls for fine rather than intermediate crushing. These changes resulted in increased capacity, a finer feed to the ball mills, and lower costs. Two additional ball mills were installed. Recleaning of concentrates in a Forrester machine was started.
1931	All Minerals Separation machines were dismantled and replaced by Forrester machines; this change resulted in the production of higher-grade concentrates, tailings of lower copper content, and decreased costs.

The tabulation which follows presents operating results and concentrator costs for the years 1923 to 1931, inclusive.

Operating results and concentrator costs for the years 1923 to 1931, inclusive

Year	Average ore milled per day, dry tons	Concentrates, per cent copper	Recovery, per cent1/	Total milling cost per ton of ore treated
1923	2,177	14.1	91.4	$0.710
1924	2,527	20.8	88.7	.565
1925	2,792	20.3	88.7	.517
1926	3,177	20.8	91.1	.463
1927	3,707	19.9	89.8	.430
1928	4,422	20.1	89.9	.395
1929	5,275	22.1	88.6	.378
1930	6,021	24.9	88.7	.303
19312/	6,228	26.6	87.5	.231

1/ Decreases in recovery are due to lower total copper content and to increased oxidized copper content of mill feed.
2/ Seven months.

COARSE CRUSHING

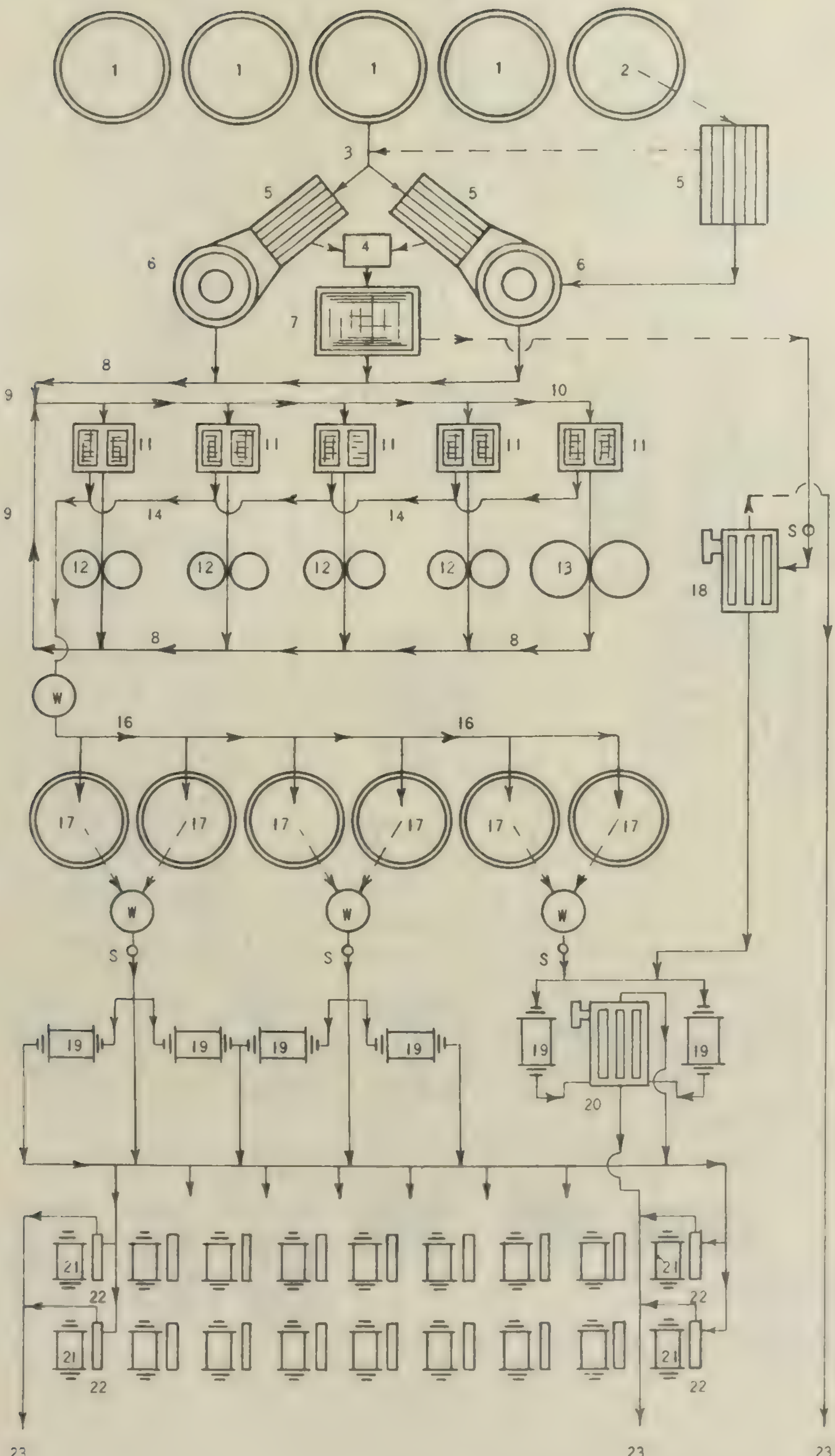
Primary crushing is done at the mine. The crushers are installed in underground chambers on the final raise through which all ore passes enroute to the concentrator. Primary crushing equipment comprises one 36 by 48 inch and one 24 by 36 inch Buchanan jaw crusher; under normal conditions only the larger machine is operated. Crushing of ore is done on three shifts, but the capacity of the larger crusher is such that actual running time is only about 50 per cent of the total.

The feed to the crusher, regulated by finger gates, passes over grizzlies with 6-inch spaces; the crusher which handles the oversize is set with a 6-inch opening. The crushed product is discharged directly into a main raise and from there delivered to the receiving bins at the concentrator.

The operation of the coarse and intermediate crushing units is complicated greatly by the condition of the ore handled; the latter is nearly always sticky and actually soupy during some seasons of the year. To cope with this problem it has been necessary to equip the receiving bins with air gates and drum-type revolving feeders and to provide means for the removal of slimes by washing and screening before proceeding further to crush the coarser portion of the ore.

PRESENT METHOD OF CONCENTRATING

Figure 1 gives the flow sheet of intermediate crushing and grinding and Figure 2 that of flotation and dewatering operations.



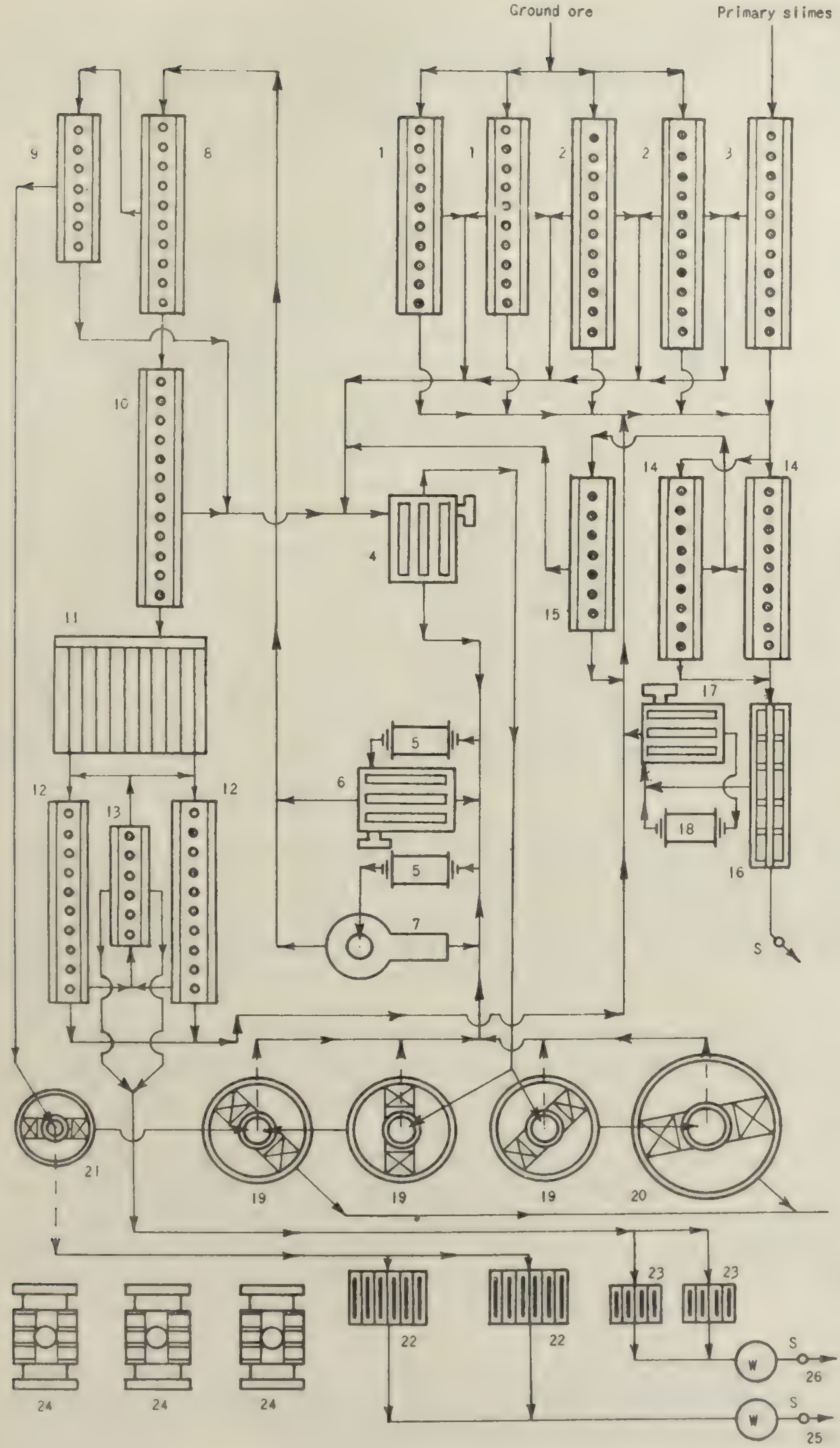
LEGEND:

1. Four coarse-ore bins, 500-ton
2. One wet-ore bin, 500-ton
3. Conveyor, 30-inch
4. Washing elevator
5. Stationary grizzlies
6. Two Symons cone crushers, 5 1/2-foot
7. Washing screens, 4 by 5 foot
8.
9. } Conveyors, 42-inch
10.
11. Five Hum-mer dry screens
12. Four Traylor rolls, 54 by 18 inch
13. One Traylor roll, 72 by 20 inch
14.
15. } Conveyors, 30-inch
16.
17. Six fine-ore bins, 600-ton
18. Drag classifier, primary slimes
19. Six primary ball mills, 6 by 11 foot
20. Drag classifier, 6-foot
21. Eighteen secondary ball mills. 5 1/2 by 9 foot
22. Eighteen Dorr simplex classifiers, 3 by 20 foot
23. Feed to flotation

W. Weightometers

S. Samplers

Figure 1.- Flow sheet of crushing and fine grinding



LEGEND:

1. Two roughers, 1 spare, 75-foot
2. Two roughers, 100-foot
3. Roughers, slimes, 100-foot
4. Drag classifier, 8-foot
5. Two concentrate regrinding mills, 7 by 10 foot
6. Drag classifier, 6-foot
7. Bowl classifier, 10-foot
8. Cleaner, 75-foot
9. Recleaner, 30-foot
10. Flotation machine treating cleaner tails, 160-foot
11. Blanket gold-recovery plant
12. Two pyrite roughers, 30-foot
13. Pyrite cleaner, 12-foot
14. Two scavenging machines, 70-foot
15. Scavenging concentrate cleaner, 50-foot
16. Hydraulic classifier
17. Drag classifier, 6-foot
18. Oversize regrinding mill
19. Three Dorr thickeners, 44 by 14 foot
20. Dorr thickener, 60 by 12 foot
21. Dorr thickener. 20 by 12 foot
22. Two American filters, 6-leaf, 8-foot diameter
23. Two American filters, 4-leaf, 6-foot diameter
24. Three Root's blowers, 1 spare
25. Copper concentrate to storage
26. Pyrite concentrate to storage

W. Weightometers

S. Samplers

Figure 2.- Flow sheet of flotation, settling, and filtering

A side elevation of the concentrator is presented in Figure 3. The building is constructed of steel and reinforced concrete, with walls and roof of corrugated iron. The erection of the plant on a site having a 45° slope permits the general use of gravity flow of pulp.

Intermediate Crushing

The minus 6-inch product from the receiving bins is delivered by a 30-inch conveyor to a washing hopper where water is added by sprays, thence over two stationary grate-type grizzlies equipped with locally cast manganese steel bars spaced with 5/8-inch openings. The grizzlies are placed in parallel and the oversize of each is delivered to one 5-1/2-foot Symons cone crusher set to crush to 5/8 inch. The grizzly undersize falls to an elevator boot and from there is elevated a distance of 30 feet and passed down a baffled washing launder to two sets of Hum-mer screens. The latter are operated wet and are equipped with screen cloth having a total surface of 15 by [illegible] feet and holes 3/16 by 1/2 inch. The screen undersize passes directly to a drag classifier for the removal of primary slimes; the sands of this classifier are delivered to the fine-grinding department. The oversize products of the wet Hum-mer screens join the Symons crusher products and are distributed to five units of Hum-mer screens and Traylor rolls; the latter operate in closed circuit with the screens. Four sets of rolls are 54 by 20 inch in size and the other set is 72 by 20 inch. Each screen unit comprises two sets of 5 by 4 foot screens equipped with cloth having 3/16 by 5/8 inch holes. The undersize of the screens is conveyed to six 600-ton capacity fine-ore storage bins at the head of the fine-grinding department.

The Symons cone crushers are belt driven at a speed of 250 gyrations per minute by 200-hp. induction motors; actual consumption of power averages about 160 hp. Manganese-steel liners are cast in the local foundry and contain 12 to 14 per cent of manganese and 1.20 per cent of carbon; cone liner consumption averages 0.022 pound per ton of ore crushed.

The manganese-steel grizzlies are of rigid grid construction. This type is very advantageous in that it does not clog because it can not spring; also it does not pass oversize material.

The rolls are heavily choke fed and are driven by individual 150-hp. motors. The 54-inch rolls operate at a speed of 100 r.p.m. and the 72-inch rolls at 83 r.p.m. Roll shells are cast in the local foundry and contain 0.50 to 0.55 per cent of carbon, 0.70 per cent of manganese, and 0.50 per cent of chromium. They are 7 inches thick as cast; roll shell steel consumption averages 0.19 pound per ton of ore handled.

Tyler screen cloth is used on both wet and dry Hum-mer screens. Although all screens are equipped with cloth having holes 3/16 inch wide, the undersize from these screens is approximately minus 5/32-inch maximum size due to partial blinding.

The intermediate crushing plant is operated an average of 20 hours per day and handles 350 tons of original feed per hour. Approximately 30 per

cent of the feed is removed by the wet screens ahead of the cone crushers. About 15 per cent of the remaining 70 per cent of the mill feed is minus 5/8-inch size, hence the two Symons cone crushers handle feed at the rate of 215 tons per hour and the combined roll circuits handle approximately 250 tons of new feed per hour. The circulating load in the roll circuits amounts to about 400 per cent of the new feed.

An important saving in maintenance is effected by turning down roll shells and other parts by means of a lathe and boring mill placed on the roll floor and under the crane. This work is done by repairmen during slack hours.

As previously noted, the washing and screening of ore ahead of the crushers and rolls is for the sole purpose of removing adhering fines which otherwise cause complete blinding of the dry operated Hum-mer screens as well as the choking of crushers and rolls. It would be impossible to maintain either capacity or fineness of product without the washing units.

The moisture content of the final intermediate crushing plant product, excluding the undersize of the wet Hum-mer screens, averages 2.5 per cent. Screen analyses of products are given in Table 1.

Fine Grinding

The feed to the fine-grinding department is distributed both to the primary and secondary ball mills; two-thirds of the original feed goes to the primary mills and the remaining one-third to the secondary.

The primary grinding units comprise six 6 by 11 foot ball mills, which are operated in open circuit; theyare driven at a speed of 24 r.p.m. by 100-hp. motors direct connected through flexible couplings.

The secondary grinding circuits comprise eighteen 5 1/2 by 9 foot Traylor ball mills; each is operated in closed circuit with a 3 by 20 foot Dorr simplex drag classifier. The ground products from the primary mills are delivered to the drag classifiers of the secondary mills with the new ore feed to the secondary mills previously noted. The classifier sands provide feed to the secondary mills, from which the ground discharge returns to the classifiers. The secondary mills are driven at a speed of 20 r.p.m. by 75-hp. motors through double reduction gears and pinions. The pulp consistency in the ball mills is about 70 per cent of solids; the classifier overflows contain about 34 per cent of solids. Classifier rakes are operated at a speed of 34 strokes per minute; the circulating load amounts to from 150 to 200 per cent.

The ball load for each mill consists of 7 tons of 3 1/2-inch balls. The latter are made at the plant from scrap rail steel, as described hereafter; consumption of ball steel averages 2 1/2 pounds per ton of ore milled.

All grinding mills are lined with scrap rail steel. The rails are cut in 7-inch lengths by a hot saw and are then hardened by quenching in cold water. The rail sections are set on end in high-strength concrete in both

the barrel and ends of the mill. The lining is initially 7 inches thick and gives approximately two years of service before being rejected; the discarded lining is from 2 to 3 inches thick. The use of rail liners on hard Britannia ore has been found to be economical, providing that the steel is hardened by heat treatment as described.

The ball mills were originally designed to operate with a pebble load; the inside diameter, therefore, has been reduced by inserting wooden lagging, 2 inches thick, between the shell and the rail liners in order to adapt the mills to a ball load.

As indicated in Figure 1, the undersize from the washing screens of the crushing plant is passed to one 7-foot drag belt classifier which removes the primary slimes for separate flotation treatment. The classifier sands unite with feed from the fine-ore bins for grinding in the primary and secondary ball mills.

Table 1, at the end of the paper, presents screen analyses of heads and finished products of the grinding units.

FLOTATION

General Description of Treatment

Referring to Figure 2, the flotation treatment comprises first the production of bulk concentrates which contain the chalcopyrite and pyrite with minor gold and silver content. The bulk concentrates are dewatered and after additional grinding are again subjected to flotation. The latter operation produces finished copper concentrate, finished pyrite concentrate, and middlings which are retreated. A blanket gold-recovery unit is provided between the cells producing copper concentrate and those producing pyrite concentrates.

The method of treatment as outlined is especially adapted to Britannia ore, in which the sulphide minerals are liberated from the gangue minerals at a much coarser size than the sulphides are liberated from each other. The depressing of the pyrite in the primary flotation circuit has been found to result in sacrifice of either recovery or grade of concentrates unless relatively fine grinding of the entire ore is practiced. With the present method of treatment a satisfactory recovery is obtainable even with a coarsely ground primary feed; the grade of concentrates is increased by additional grinding and floating of the bulk concentrates, thus avoiding the expense of fine-grinding the hard gangue material. The feed to the concentrate regrinding plant amounts to only 12 to 15 per cent of the total ore milled. A further advantage in the present method of treatment lies in the fact that it is not necessary to maintain selective flotation conditions between the chalcopyrite and pyrite in the primary circuit; this results in a greater copper recovery, as stronger reagent combinations may be used.

Flotation Equipment

Forrester machines which have a combined length of 977 feet comprise the flotation equipment. These machines are utilized as indicated in the following tabulation.

Description and use of Forrester machines

Number	Length of each machine, feet	Operation for which used
2	100	Roughing ball-mill products.
2	75	Roughing ball-mill products; one is a spare.
1	100	Roughing primary slimes.
2	70	Scavenging rougher-machine tailings.
1	50	Cleaning scavenger-cell concentrates.
1	75	Floating the chalcopyrite from the reground bulk concentrates.
1	160	Floating chalcopyrite-pyrite middlings from bulk concentrates.
1	30	Cleaning copper concentrate.
2	30	Roughing pyrite concentrate.
1	12	Cleaning pyrite concentrate.

All Forrester machines are 47 inches wide and 36 inches deep. They are equipped with 1/2-inch air pipes spaced 3 inches apart except at the head ends of the roughers where the spacing is 2 1/2 inches for the first 20 feet. Air is supplied by two Connersville blowers, each of 124 cubic foot displacement per revolution; one is driven at a speed of 190 r.p.m. by a 300-hp. synchronous motor and the other at 160 r.p.m. by a 200-hp. induction motor. Air consumption is 45 cubic feet of free air per minute per linear foot of machine; air pressure at the blower is 2 pounds per square inch and at the machines is 1 3/4 pounds. The actual rate of power consumption by flotation machines amounts to 450 hp., which is equivalent to 1.24 kw.-h. per ton of ore milled.

Reagents

Flotation is conducted in a lime, xanthate, and pine oil circuit; a small quantity of No. 25 aerofloat is also used. The amounts of reagents used and the points of addition are indicated in the following tabulations.

Details of reagents used

Reagent	Amount used per ton of ore milled, pounds	Where added
Hydrated lime ...	0.60	At head of each fine grinding mill.
Potassium ethyl xanthate	.01	At head of each rougher treating ball-mill pulp.
Pine oil	.05	
Xanthate	.008	At center of each rougher treating ball-mill pulp.
Pine oil	.035	
Xanthate	.01	At head of rougher treating primary slimes.
Pine oil	.03	
Lime	.40	To bulk concentrate pulps entering thickeners.
Lime	.10	At head of each concentrate regrinding mill.
Pine oil	.01	At head of machine treating reground concentrates.
Pine oil	.005	At head of copper concentrate cleaner.
Pine oil	.005	At head of the 160-foot machine handling reground concentrates.
Xanthate	.002	
Aerofloat, No. 25	.005	
Pine oil	.005	At center of the 160-foot machine handling reground concentrates.
Pine oil	.01	At heads of pyrite roughers.
Xanthate	.01	

Summary of reagents used

	Total used per ton of ore milled, pounds
Hydrated lime	1.10
Potassium ethyl xanthate	.04
Aerofloat, No. 25	.005
Pine oil	.15

Lime is fed by means of a slowly moving conveyor belt; the rate of addition is regulated by the depth of lime on the belt. Water is added to the lime in a mixing tank; the milk of lime is distributed from the latter to the various points of addition by launders and pipe lines. Other reagents are added to the pulps by Callow feeders.

The amount of lime added at the ball mills is sufficient to maintain a very slight degree of alkalinity in the rougher tailing pulp; the pH value of this pulp is between 8.0 and 8.5. The lime added to the thickeners aids the settling of solids and also acts as a depressor of pyrite in subsequent flotation operations. The alkalinity of the thickened pulp is increased slightly by a further addition of lime to the concentrate regrinding mills. The pH value in the selective flotation circuit is maintained between 10.0 and 10.5.

The very satisfactory retardation of the pyrite and the consequent high grade of copper concentrate produced in selective flotation is obtained by the long time of conditioning of the bulk concentrates with lime in the thickeners and regrinding mills and by careful control of reagents. It is noted that xanthate is the only reagent used to reactivate the pyrite; the amount used for this purpose is approximately 0.40 pound per ton of pyrite concentrates recovered.

Primary Slimes

As previously noted, the primary slimes are separated from the feed by screens. The screen undersize is treated in a drag classifier and the overflow from the latter is treated in a separate 100-foot rougher machine. The ore contains appreciable quantities of soluble salts, which segregate with the primary slime pulp. The amount of salts varies considerably with the season of the year. The tabulation which follows gives an average analysis of the water decanted from the slime feed pulp.

Soluble salts contained in Britannia ore

	Grams per liter	Pounds per ton of slimes	Pounds per ton of mine ore
Copper sulphate	0.12	1.2	0.17
Iron sulphates	.23	2.3	.33
Free acid	.06	.6	.09
Calcium sulphate	1.20	12.0	1.71
Magnesium sulphate	.10	1.0	.14

Nearly all of these constituents have been found to be harmful to flotation operations; calcium sulphate is especially troublesome and is present in the largest amount. Until near the end of 1930, soda ash was added regularly as a modifying reagent and in this role precipitated most of the soluble salts. As the amount of soluble salts in the ore steadily increased, the amount of soda ash required reached a quantity almost prohibitive from the standpoint of expense. In present practice, therefore, neutralizing reagents are not used, but the time of flotation treatment has been lengthened to give a better opportunity for the minerals which float slowly to be recovered.

The tailings from the treatment of primary slimes contain from 0.20 to 0.30 per cent of copper as compared to 0.07 to 0.10 per cent copper content for the remaining flotation tailings. At Britannia, there is no question as to the advantage of separate treatment of primary slimes, comparative tests from time to time having demonstrated that the combined treatment yields final mill talings which contain at least 0.05 per cent more copper than the tailings from the present method of treatment.

SELECTIVE FLOTATION OF BULK CONCENTRATES

All bulk flotation concentrates including those from the treatment of primary slimes and those from scavenging operations are delivered to one 8 by 18 foot drag belt classifier. The underflow goes directly to the concentrate regrinding mills and the overflow to four Dorr thickeners; three of the latter are 44 feet and the other 60 feet in diameter. The thickener overflow is clear and is wasted; the thickened pulps which contain about 55 per cent of solids are sent to the concentrate grinding mills.

Two 7 by 10 foot ball mills are used for grinding concentrates; one mill operates in closed circuit with a 6-foot drag belt classifier and the other in closed circuit with a 10-foot Dorr bowl classifier. Ball rejects, 1 1/2 inch in diameter or less, from the fine grinding mills are used as the grinding media. The mills function as conditioners as well as grinders.

The classifier overflow pulps which contain 20 per cent of solids, the latter being ground 70 to 80 per cent minus 200-mesh, are pumped to one 75-foot Forrester machine. The froth is cleaned in one 30-foot recleaner cell; the tailings are treated further in one 160-foot machine which produces middlings and tailings. The middlings, with the middlings from the recleaner cell, are returned to the drag classifier at the head of the regrinding unit. The tailings of the 160-foot machine consist mainly of pyrite but also contain a small amount of relatively coarse and difficultly floatable free gold. The tailings are delivered, therefore, to a blanket plant for the recovery of this gold, after which the pyrite is floated in two 30-foot rougher machines, the rougher concentrate being cleaned in one 12-foot machine.

Table 1 gives screen analyses of flotation heads, bulk concentrates, and reground bulk concentrates; Table 2 presents chemical analyses of intermediate and final flotation products. Table 3 gives screen-assay analyses of finished copper concentrate and pyrite concentrates; Table 4 presents screen-assay analyses of rougher sand tailings, slime tailings, and final mill tailings.

Blanket Plant

The equipment provided for the recovery of gold in the blanket plant consists of seven shallow parallel launders 2 1/2 feet wide and 40 feet long; they are set with a grade of 1 1/2 inches to the foot. Medium-grade woolen blankets, held in place by loose iron strips at the sides, are used to collect the gold. The pulp flows over the blankets in an even stream about 1/2 inch deep and the grade is sufficient to prevent banking. Blankets are picked up

on rollers and washed, concentrate side downwards, in a small tank once in every eight hours; they are discarded after about three months of service. The concentrates assay from 3 to 5 ounces of gold per ton and contain approximately 10 per cent of the total gold in the heads.

SEPARATION AND RETREATMENT OF COARSE MATERIAL IN THE TAILINGS

In 1928 a 7-compartment hydraulic classifier was installed to recover plus 35-mesh material from the rougher tailings. The spigot products from this classifier were reground in a separate grinding mill and returned for flotation treatment as indicated in Figure 2. This practice was instituted at a time when the mill grinding was exceedingly coarse and the copper loss in the oversize material amounted to a large percentage of the total. However, with recent improvements and additions made in the crushing and grinding departments, the feed to the roughing machines is now considerably finer and the possible recovery obtainable in the hydraulic classification plant is consequently much lower. This plant is, therefore, not being operated at present, as the additional expense is not warranted, especially in view of the current low value of the copper recovered.

DEWATERING AND HANDLING OF FINISHED CONCENTRATES

Finished copper concentrate is thickened in a 20-foot Dorr thickener and the thickened pulp is filtered by two 8-foot diameter, 6-leaf American filters. Pyrite concentrate is filtered directly by two 6-foot diameter, 4-leaf American filters. Usually two filters, one for copper and one for pyrite, are sufficient to handle the concentrates produced. Vacuum at the filters is maintained at 25 inches of mercury by two Ingersoll-Rand pumps which have 22-inch cylinders and operate with 8-inch strokes.

The copper and iron concentrates are conveyed from the filters, a distance of approximately 1,000 feet, to storage bins by 16-inch and 12-inch belts. The capacities of storage bins are 6,000 tons and 4,000 tons for copper and pyrite concentrates, respectively. Concentrates are shipped by steamer, the copper product going to the Tacoma smelter. Loading of steamers is by means of a gantry crane and a clamshell bucket which discharges the material into a hopper; the latter is equipped with an apron feeder which delivers to a 24-inch conveyor belt. The conveyor belt passes over a Merrick weightometer and then discharges directly into the hold of the vessel. The outer end of the conveyor is hinged so that adjustment can be made for the depth of the vessel and the stage of the tide. The averaging loading rate is about 175 tons per hour and the cost of handling is $0.04 per ton loaded.

DISPOSAL OF TAILINGS

Tailings are conveyed by launder and pipe line and discharged into tidewater. Water is not reclaimed from the tailings pulp.

SAMPLING AND METALLURGICAL CONTROL

Dry ball-mill feed for each section is weighed by a Merrick weightometer on the conveyor belt enroute from the fine-ore storage bins to the grinding units. The weight of wet feed screened out in the crushing plant is estimated by means of a factor applied to the weight of ore passing through the crushing plant, which weight is determined by a weightometer placed between the roll sections and the fine-ore bins.

A sample of concentrator heads to each of the three mill sections is taken by an automatic cutter placed between the fine-ore bins and the grinding units. These cutters are actuated by air cylinders from a master valve, which in turn is operated by a water tipple. The same master valve also controls an air-operated tailings sampler placed on the main launder which conveys all tailings to waste. Samples of concentrates are taken hourly by hand from the belts conveying the copper and pyrite concentrates to their respective storage bins.

Several years ago a small laboratory was established adjacent to the flotation floor in the concentrator, and a system of shift sampling and assaying was instituted in order to obtain close and accurate control of the performance of the various flotation units. Samples of the individual rougher machine tailings of the bulk flotation circuits and samples of the rougher, cleaner, and iron tailings of the selective flotation circuit are taken three times per shift and rapidly dried on a 4 by 8 foot electric hot plate. Rougher tailings of the bulk flotation circuits are assayed by the colorimetric method and higher grade products by a fluoride modification of the iodide method; results are available about 45 minutes after samples are taken.

Alkalinity of pulps is controlled by titration of tailings water with weak sulphuric acid solution using phenol as an indicator. Periodical determinations of pH values are also made.

General shift samples are assayed in the main Beach laboratory where colorimetric and standard iodide methods are used. Smelter control samples are assayed by both iodide and electrolytic methods, one being used as a check on the other. Moisture determinations on concentrates are made at the concentrator laboratory immediately after the samples are collected at the end of each shift. The method used for determining oxide copper content of products consists in leaching with cold 5 per cent sulphuric acid for one hour with occasional stirring; the copper content of the filtrate is determined by electrolysis.

Metallurgical data for the month of July, 1931, are given in Table 5.

LABOR

The supervising staff consists of the superintendent, the metallurgist, and the general crushing plant and repair foremen. The crushing plant is operated by a shift boss, three operators, and five helpers per shift. The grinding and concentrating units are handled by a shift boss, three operators, and three helpers per shift; one operator is employed on ball mills, one for

flotation machines, and the other for filters. The flotation operator runs the shift assays and collects and dries the automatic samples. An oiler is employed in both the mill and crushing plant, and a reagent mixer is employed in the mill on day shift only.

Repairs are carried out by a foreman and a crew of five men in the crushing plant and by a foreman and one man in the mill. One electrician and a helper are responsible for electrical maintenance.

Two men are assigned to research and testing work and two assayers handle routine samples and control assays.

It is worthy of note that the present low cost of milling has not been achieved by recent reductions in the number of operators, or by any reduction in wages, although the replacement of Minerals Separation by Forrester machines has made possible the elimination of one man a shift from the flotation department.

CONVEYING

A specially constructed 9-ply, 30-inch belt with 3/16-inch cover and 1/8-inch back, equipped with breaker strips and reinforced edges, is used for the very hard service of conveying the soupy and sticky ore from the feeders under the coarse-ore bins to the cone crushers.

The circulating loads in the roll circuits are handled by 7-ply 42-inch conveyors which also elevate the ore to the Hum-mer screens above the rolls. A 30-inch belt delivers the crushed product from the screens to the fine-ore bins. The only elevator used is that employed for the washing operation in the crushing plant.

Three conveyor belts, with a slope of 20°, carry the ore from the fine-ore bins and over weightometers. From here gravity flow is utilized to the ball mills and thence to flotation.

Bulk flotation concentrate pulps flow by gravity to the thickeners and thence to the grinding mills. The reground concentrates are delivered to the head of the selective flotation circuit by a 6-inch Wilfley pump. Several short air lifts are used to handle intermediate products such as copper middling and iron tailing.

Conveyor belting renewals are charged against a reserve account for which a uniform charge of $0.002 per ton of ore milled is made.

The crushing plant is served by a 30-ton capacity traveling crane; the eighteen ball mills and accessory classifiers are provided with a similar crane placed over the two secondary mill floors. The primary mill floor is equipped with a 10-ton capacity hand crane. All floors are served by an incline railway and hoist.

FOUNDRY WORK AND GRINDING BALL MANUFACTURE

A very important factor in the achievement during July, 1931, of a milling cost as low as $0.177 per ton of ore milled is the local manufacture of all supplies used in crushing and grinding operations.

The foundry is equipped with an electric furnace and a small cupola. The cupola is used in conjunction with the electric furnace for the production of manganese-steel castings, such as cone-crusher liners, jaw plates, grizzly bars, and numerous other wearing parts. Roll shells also are made at a cost of less than $0.03 per pound. In general, our foundry experience with the manufacture of chrome roll shells and manganese cone liners has been very satisfactory and few breakages have occurred; less, in fact, than with imported castings.

Grinding balls are made in a special plant having a capacity of 40 tons of 3-inch to 3 1/2-inch balls per 8-hour day. The raw material used is scrap rail, from which finished balls are produced at a total cost well below that obtainable by any other method. The plant consists of two oil-fired furnaces, in which batches of thirteen 80 to 100 pound rails cut in 11-foot lengths are heated to a white heat; the heated rails pass to three stands of rolls, which compress the heated rails to rounds, and thence to the ball-forming machine.

The Britannia ore is very hard and exceptionally resistant to abrasion. ~~It is~~ this reason, rather than because of any inferior quality of the rail-steel balls, that accounts for the rather high ball consumption of 2.5 pounds per ton of ore milled.

The advantages of a foundry in connection with the mill are numerous. In fact, at any mill, where scrap is accumulating for lack of market, an electric furnace of a size in harmony with that of the milling operation should be of material benefit in reducing the cost of milling.

MILLING COST

The Britannia mill operation is favored with a low unit cost of power and a simple method of tailings disposal. It is, however, burdened with an extraordinary expense of handling, under pressure, soupy mine ore received in several months of the year; with a washing process in connection with the crushing plant operation; and with high power consumption and costly maintenance in crushing and grinding operations owing to the extremely hard ore handled. The adverse factor of hard ore is to some extent offset by the milling practice adopted in which relatively coarse primary grinding is not inconsistent with satisfactory metallurgical results.

Table 6 presents a summary of milling costs for the month of July, 1931; Tables 7 and 8 give distributions of labor and power, respectively, for the same period.

Table 1. - Screen analyses of intermediate concentrator products for July, 1931[1]

Screen size	Run-of-mine	Jaw-crusher discharge	Symons cone discharge	Per cent: Ball mill feeds: Dry feed	Ball mill feeds: Wet feed	Flotation feeds: Sands	Flotation feeds: Primary slimes	Bulk flotation concentrates	Reground bulk flotation concentrates
Plus 1-inch, square			Nil	-	-	-	-	-	-
Plus 7/8-inch, square			1.5	-	-	-	-	-	-
Plus 3/4-inch, square	36-inch maximum size	6-inch maximum size	10.0	-	-	-	-	-	-
Plus 1/2-inch, square	-	-	19.5	-	-	-	-	-	-
Plus 4-mesh...	-	-	56.0	3.8	3.5	-	-	-	-
Plus 6-mesh...	-	-	2/13.0	19.2	10.2	-	-	-	-
Plus 8-mesh...	-	-	-	20.9	10.4	-	-	-	-
Plus 10-mesh..	-	-	-	16.6	11.0	-	-	-	-
Plus 14-mesh..	-	-	-	9.6	8.2	-	-	-	-
Plus 28-mesh..	-	-	-	11.2	11.2	3.0	0.9	-	-
Plus 35-mesh..	-	-	-	2.4	4.0	6.0	3.0	-	-
Plus 48-mesh..	-	-	-	2.7	4.4	11.0	4.5	-	-
Plus 65-mesh..	-	-	-	2.2	3.6	11.5	6.0	4.0	
Plus 100-mesh.	-	-	-	3/11.4	3/33.5	12.0	9.1	10.0	3.0
Plus 150-mesh.	-	-	-	-	-	9.0	10.8	14.0	10.0
Plus 200-mesh.	-	-	-	-	-	8.0	7.3	11.0	11.0
Minus 200-mesh	-	-	-	-	-	39.5	58.4	61.0	76.0
Total	-	-	100.0	100.0	100.0	100.0	100.0	100.0	100.0

1/ Screen analyses of finished concentrates and tailings are given in Tables 3 and 4, respectively.

2/ Minus 4-mesh.

3/ Minus 65-mesh.

Table 2. - Chemical analyses of intermediate and final flotation products for July, 1931 1/

	Per cent							Ounces per ton	
	Moisture	Copper		Iron	Zinc	Sulphur	Insoluble	Gold	Silver
		Total	Oxide						
Bulk concentrates - sand flotation sections ...	-	9 to 10	-	-	-	-	11 to 14	-	-
Bulk concentrates - slime flotation section	-	10 to 11	-	-	-	-	15 to 20	-	-
Feed to selective flotation circuit	-	10 to 11	-	-	-	-	-	-	-
Rougher concentrates - selective flotation circuit	-	24 to 26	-	-	-	-	-	-	-
Tailings, first rougher cell, selective circuit	-	3 to 4	-	-	-	-	-	-	-
Rougher middlings, selective circuit	-	8 to 12	-	-	-	-	-	-	-
Cleaner middlings, selective circuit	-	6 to 10	-	-	-	-	-	-	-
Rougher tailings, sand sections	-	0.07 to 0.10	0.010	1.5 to 2.0	-	-	-	-	-
Rougher tailings, slime section	-	.16 to .30	.050	2.0 to 3.0	-	-	-	-	-
Final iron tailing, selective circuit	-	.20 to .30	-	35 to 40	-	-	-	-	-
Final copper concentrate	10.6	27.63	.100	30.8	2.8	36.4	2.0	0.06	2.45
Final pyrite concentrate	8.2	0.32	-	44.2	-	50.8	3.1	.01	.30
Final mill tailings	-	0.11	.015	3.3	-	1.3	-	.001	.03

1/ Approximate upper and lower limits are given for assays of intermediate products; assays of final products given are actual monthly averages.

Table 3. - Screen-assay analyses of finished concentrates

Copper Concentrate

Screen size, mesh	Weight per cent	Copper, per cent		Iron, per cent		Insoluble, per cent	
		Assay	Of total	Assay	Of total	Assay	Of total
Plus 150 ...	4.3	29.08	4.5	31.1	4.3	1.8	3.7
Plus 200 ...	8.4	28.86	8.7	31.4	8.5	1.4	5.6
Minus 200 ..	87.3	27.63	86.8	30.8	87.2	2.2	90.7
Total, or average ..	100.0	27.73	100.0	30.9	100.0	2.1	100.0

Pyrite Concentrates

Screen size, mesh	Weight per cent	Copper, per cent		Sulphur per cent		Insoluble, per cent	
		Assay	Of total	Assay	Of total	Assay	Of total
Plus 100 ...	1.7	1.40	9.0	49.2	1.6	5.1	2.8
Plus 150 ...	14.0	0.45	23.9	50.3	13.9	3.2	14.7
Plus 200 ...	19.2	0.24	17.5	50.9	19.3	2.9	18.3
Minus 200 ..	65.1	0.20	49.6	50.7	65.2	3.0	64.2
Total, or average ..	100.0	0.26	100.0	50.6	100.0	3.0	100.0

Table 4. - Screen-assay analyses of tailings for July, 1931

Screen size, mesh	Rougher sand tailings, per cent			Rougher slime tailings, per cent			Final mill tailings per cent		
	Weight	Assay, copper	Of total copper	Weight	Assay, copper	Of total copper	Weight	Assay, copper	Of total copper
Plus 28..	4.0	0.08	3.2	1.0	0.14	0.6	3.5	0.09	2.8
Plus 35..	6.6	.09	5.9	3.2	.18	2.4	6.4	.10	5.6
Plus 48..	13.4	.11	14.7	4.8	.24	4.9	12.6	.11	12.2
Plus 65..	12.0	.09	10.8	6.4	.31	8.5	10.5	.10	9.3
Plus 100.	13.2	.08	10.5	10.0	.30	12.8	12.7	.09	10.1
Plus 150.	7.6	.07	5.3	10.4	.26	11.5	7.7	.08	5.4
Plus 200.	5.2	.07	3.6	6.8	.19	5.5	5.7	.07	3.6
Minus 200	38.0	.12	46.0	57.4	.22	53.8	40.9	.14	51.0
Total, or average	100.0	0.099	100.0	100.0	0.234	100.0	100.0	0.112	100.0

Table 5. - Metallurgical data for July, 1931[1/]

Total ore treated	tons	201,065
Period of operation	days	31
Average ore treated per 24 hours	tons	6,486
Total copper concentrate produced	do.	6,503
Average copper concentrate produced per 24 hours	do.	210
Total pyrite concentrate produced	do.	2/ 4,650
Average pyrite concentrate produced per 24 hours	do.	150
Recovery of total copper	per cent	89.36
Recovery of sulphide copper	do.	90.71
Ratio of concentration for copper concentrate	tons into 1	30.9
Water consumption per ton of ore milled	imperial gallons	1,100
Ball consumption per ton of ore milled	pounds	2.50
Liner consumption per ton of ore milled	do.	.18
Consumption of reagents per ton of ore milled:		
Hydrated lime	do.	1.10
Pine oil	do.	.15
Potassium xanthate	do.	.04
Aerofloat	do.	.005
Power consumption per ton of ore milled	kilowatt-hours	12.84

1/ On account of the present production curtailment policy, the mill did not operate full time during July. The above production figures for a month have, therefore, been calculated by using an average day's operation as the basis.

2/ Does not indicate maximum possible recovery.

Table 6. - Summary of milling costs for July, 1931

	Operating				Repairs		Total
	Labor	Power1/	Supplies	Reagents	Labor	Supplies	
Breaking to 6-inch size	$0.0023	$0.0002	$0.0020	--	$0.0005	$0.0002	$0.0052
Intermediate crushing	.0178	.0050	.0158	--	.0063	.0037	.0486
Fine grinding and classifying	.0064	.0148	.0177	--	.0040	.0031	.0460
Flotation	.0030	.0031	--	$0.0167	.0028	.0021	.0277
Dewatering and handling concentrates	.0027	.0015	.0012	--	.0018	.0022	.0094
Sampling and assaying	.0017	--	.0003	--	--	--	.0020
Superintendence, office and warehouse	.0037	--	.0005	--	--	--	.0092
Miscellaneous	.0074	.0004	.0072	--	.0031	.0008	.0189
M. S. royalty	--	--	.0100	--	--	--	.0100
Totals	0.0500	0.0250	0.0547	0.0167	0.0185	0.0121	0.1770

1/ Power cost was $0.0020 per kw.-h. for July, 1931. On account of July being a flush water month, nearly all power consumed was generated by the company's own hydroelectric plant, hence the low unit cost for power. The average year-round cost for power is approximately double the above figure, which would then give an average milling cost of approximately $0.20 per ton of ore milled.

Table 7. - Distribution of labor for July, 1931

	Ore handled per 8-hour man-shift, tons		
	Operating	Repairs	Total
Breaking	2,236	-	-
Intermediate crushing	223	-	-
Fine grinding and classifying	1,081	-	-
Flotation	1,622	-	-
Dewatering and handling concentrates	1,622	-	-
Concentrator foremen	2,162	-	-
Pyrite production	2,162	-	-
Miscellaneous	1,622	-	-
Experimental	6,486	-	-
Assaying and sampling	3,243	-	-
Superintendence	1,622	-	-
Average	108	295	80

Table 8. - Distribution of power for July, 1931

	Power consumption per ton of ore milled, kilowatt-hours	Per cent of total power
Breaking	0.10	0.8
Intermediate crushing	2.60	20.3
Fine grinding and classifying	6.84	53.3
Regrinding of concentrates	.97	7.5
Flotation	1.44	11.2
Dewatering and handling concentrates	.71	5.5
Miscellaneous	.18	1.4
Total	12.84	100.0

Figure 3.- Sectional elevation of mill

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

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GEOPHYSICAL ABSTRACTS

NO. XXXVI

BY

Frederick W. Lee

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GEOPHYSICAL ABSTRACTS[1]

No. 36

Compiled by Frederick W. Lee[2]

List of contributing editors of Geophysical Abstracts:

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Barton, Dr. D. C., Petroleum Building, Houston, Tex.
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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6620."

2 Senior physicist, U. S. Bureau of Mines.

1. GRAVITATIONAL METHODS

(694) DIE BEDEUTUNG DER FINNISCHEN SCHWEREMESSUNGEN FÜR DIE ANGEWANDTE GEOPHYSIK

(THE IMPORTANCE OF FINNISH GRAVITY MEASUREMENTS FOR APPLIED GEOPHYSICS)

By H. Reich

Gerlands Beiträge zur Geophysik, Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 2, No. 1, 1931, pp. 1-13.

The possible causes of the Finnish gravimetric disturbances, as made known by U. Pesonen, are discussed. The distance of the regions of the Rapakivi granites and of the Central granites, as well as the Graben-regions with broken tectonics usually has a negative sign. Positive disturbances are registered along the line of magmatites of the old post-Bothnic (archean) fold rocks. In addition to these local anomalies the gravity picture is influenced, it seems, by the regional isostatic movements (ice-isostatics). The practical importance of these facts for the interpretation of gravity anomalies is emphasized.--Author's abstract translated by W. Ayvazoglou.

(695) ERGEBNISSE VON SCHWEREMESSUNGEN BEI DORSTEN (WESTFALEN)

(RESULTS OF GRAVITY MEASUREMENTS NEAR DORSTEN (WESTPHALIA))

By O. Barsch

Gerlands Beiträge zur Geophysik, Ergänzungshefte für angewandte Geophysik, Leipzig, vol. 2, No. 1, 1931, pp. 14-21.

The author makes an attempt to establish, by means of torsion-balance measurements, the more or less important disturbances in the Rhine-Westphalia coal basin manifested from about 500 to 600 meters below the younger overburden, as well as to mark their limits.

In this preliminary work the author discusses the possibility of such a determination without knowing the specific gravity difference. Based on the torsion-balance investigations it was shown that these disturbances could be clearly recognized from the map of gradients.--Author's abstract translated by W. Ayvazoglou.

(696) PENDEL UND SCHWINGUNGSDAUER-BEOBACHTUNGSVERFAHREN

(PENDULUM METHOD AND METHOD OF OBSERVING THE PERIOD OF OSCILLATION)

By O. Meisser

Gerlands Beiträge zur Geophysik, Ergänzungshefte für angewandte Geophysik, Leipzig, vol. 2, No. 2/3, 1931, pp. 131-204.

This is the first part of the article published in the same number of the "Ergänzungshefte" under the common title "Beobachtungsverfahren und Apparaturen für sehr genaue relative Schwere-und Zeitmessungen" (method of observation and apparatus for very accurate relative gravity and time measurements). The contents of this part are as follows:

1. Sphere of application of the pendulum.
2. The geometric form of the pendulum.
3. The minimum pendulum.
4. Movable masses on the pendulum.
5. Adjustment of pendulums.
6. The knife-edge of the pendulum.
7. The temperature coefficient.
8. The method of observation of the period of oscillation.
9. The pendulum apparatus.

Forty-six figures are added.--W. Ayvazoglou.

(697) DAS PHOTOGRAPHISCHE KOINZIDENZVERFAHREN UND DAS PENDEL ALS ZEITMESSER

(PHOTOGRAPHIC COINCIDENCE METHOD AND THE PENDULUM AS A TIME METER)

By H. Martin

Gerlands Beiträge zur Geophysik, Ergänzungshefte für angewandte Geophysik, Leipzig, vol. 2, No. 2/3, 1931, pp. 204-265.

This is the second part of the article published under the common title "Beobachtungsverfahren und Apparaturen für sehr genaue relative Schwere-und Zeitmessungen" (method of observation and apparatus for very accurate relative gravity and time measurements).

Contents of this part:

1. Physical methods of measurement for comparing two periods of oscillations.
2. Generalization of the oscillation method by the coincidence method.
3. Apparatus for automatic recording of coincidences.
4. Evaluation of coincidence curves recorded photographically.

5. Continuous comparison of the time of oscillation of an astronomic pendulum-clock with the time of oscillation of a free-swinging pendulum.
6. Comparison of the time of oscillation of a tuning fork with that of a free-swinging pendulum.
7. Amplitude correction.
8. Influence of a magnetic field upon the time of oscillation of a magnetizable pendulum.
9. Direct determination of the time of oscillation of a tuning form by means of coincidences.
10. The swinging pendulum as a time meter.

The author concludes that, based on a series of examples, he could prove that the pendulum and the photographic coincidence method, in various combinations, by using simple means, are suitable for carrying out time measurements of the highest precision.--W. Ayvazoglou.

(698) DAS FREIE PENDEL ALS ZEITNORMALE ÄUSSERSTER PRÄZISION

(THE FREE PENDULUM AS A TIME-STANDARD OF HIGHEST PRECISION)

By Th. Gengler

Gerlands Beiträge zur Geophysik, Ergänzungshefte für angewandte Geophysik, Leipzig, vol. 2, No. 2/3, 1931, pp. 266-307.

This is the third part of the article "Beobachtungsverfahren und Apparaturen für sehr genaue relative Schwere-und Zeitmessungen (Method of observation and apparatus for very accurate relative gravity and time measurements).

Contents of the third part:

1. Accuracy in the determination of the unit of time reached so far.
2. Astronomic precision clocks.
3. Application of a free pendulum for time measurements and corrections of it.
4. Chronographs of high precision.
5. Contact arrangement free of mechanical connection.
6. Electrical light-contact for manifold application.

The fourth part of the common article mentioned above contains a list of books (93) relating to the subject discussed.--W. Ayvazoglou.

(699) PARTICOLARI ASPETTI GRAVIMETRICI DI ALCUNI NUCLEI SUBPADANI

(SPECIAL GRAVIMETRICAL ASPECTS OF SOME NUCLEI IN THE LOWER PLAIN OF THE PO)

By A. Belluigi

Gerlands Beiträge zur Geophysik, Ergänzungshefte für angewandte Geophysik, Leipzig, vol. 2, No. 2/3, 1931, pp. 308-316.

The author gives new formulas for the calculation of the gradient and curvature with the Eötvös balance; the density differences are considered to be not constant, but a linear or quadratic or cubic function of these differences is supposed. The calculations are made with cylindrical and rectangular coordinates for a 2 or 3 dimensional space. In the boreholes in the plain of the Po (Italy) the density was found to increase with the depth for the same stratigraphical formation.--W. Ayvazoglou.

2. MAGNETIC METHODS

(700) RESULTS OF SOME MAGNETIC MEASUREMENTS ON DIKES, WITH EXPERIMENTS UPON GEOPHYSICAL DIFFERENTIATION OF NICKEL-ORE DEPOSITS IN THE SUDBURY DISTRICT, ONTARIO, CANADA

By F. W. Lee

U. S. Department of Commerce, Bureau of Mines, Washington, D. C., Technical Paper 510, 1932, 18 pp.

The purpose of this paper, according to the author, is to give the results of experimental investigations in the Sudbury region, Ontario, Canada; it may be considered a continuation of Information Circular 6235, Comparative Advantages of Applying Several Geophysical Methods of Prospecting to the Same Territory (see Geophys. Abs. 13).

In this work attention is directed primarily to distinguishing a nickel orebody or vein from other magnetic material existing in the same locality, and some magnetic observations on dikes are given.

Contents of the article:

Mineralogical and geological factors influencing geophysical investigations.
Description of ore vein.
Magnetic measurements.
Mathematical relations concerning dike anomalies: (1) Narrow dikes; (2) magnetic effects arising from various portions of dike; (3) magnetic intensities measured along traverse line over dike; (4) magnetic relations of dikes of considerable width.
Basis for graphic construction of anomalies.
Effect of topography upon magnetic anomalies.
Results of field observations.
Electrical induction methods: (1) Description of apparatus; (2) results of observations with exciting coil vertical; (3) exciting coil horizontal.

The article is provided with 20 illustrations. Conclusions reached by the author read as follows:

> It may therefore be concluded that the diabase dike may be easily discovered magnetically but offers no reaction to the induction methods. On the other hand, the ore deposit responds equally well to both magnetic and electrical methods of detection. It is therefore unnecessary to follow a diabase dike, as was done with a magnetometer, until it has outcropped to establish its identity.
>
> Very simple graphic relations exist for determining the vertical and horizontal variation of magnetic intensities over the dikes. Even upon very rough premises it is possible to construct a theoretical relation that approximates the observed values closely.
>
> In the determination of depth, more accurate results were obtained where a platform was used than could be computed from the various points on the magnetic curves.

The article is concluded by referring to the bibliography on the subject. --W. Ayvazoglou.

(701) ADDITIONAL NOTES ON OBSERVATIONS OF MAGNETIC ANOMALIES WITH A MAGNETOMETER

By J. G. Koenigsberger

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No. 4, 1931, p. 374.

This additional note concerns Koenigsberger's article published in No. 3, vol. 36, of Terrestrial Magnetism and Atmospheric Electricity (see Geophys. Abs. 33, p. 333). A graph with another abscissae-axis is given, to take the place of that in the previous article (fig. 5).

The curvatures of the profiles for the differences of H and Z against the normal values shown in this graph indicate the depth of the center of the disturbing rock on the first 25 km. of the line to be about 7 to 12 km. between kilometers 15 and 20 for the principal anomaly, and that of a second anomaly to be about 5 km. in the same region between kilometers 15 and 20 -- more probably caused by intrusive basic rocks than tectonics. The same or similar anomaly was detected on the other side of the Rhine by J. Jung and C. Alexanian (Annales Office National des Combustibles Liquides, vol. 6, No. 1, 1931; see Geophys. Abs. 30, p. 245).--W. Ayvazoglou.

(702) CONTRIBUTION A L ÉTUDE DES FAILLES EN PROSPECTION MAGNÉTIQUE

(CONTRIBUTION TO THE STUDY OF DIKES IN MAGNETIC PROSPECTION)

By J. Jung and C. Alexanian

Annales de l'Office National des Combustibles liquides, Paris, vol. 6, No. 4, 1931, pp. 711-720.

It is generally accepted that a curve calculated from the magnetic susceptibilities of diked terrains represents the anomaly of the vertical component of the terrestrial magnetic field above the dikes.

By the study of a number of dikes on the terrain it is shown that this does not agree with the theoretic curve, and that in fact more complex phenomena form the preponderant part of the anomaly. In the present article examples are given, and conclusions which are to be taken into consideration during the prospection are drawn. The translation of the authors' conclusions reads as follows:

It is recognized that from the theoretical viewpoint the problem to be solved is a very delicate one. The conclusions are thus limited to the following three statements:

1. From the magnetic viewpoint a dike can not be considered only a geometric boundary of two separate parts of terrains with different magnetic susceptibility; it produces also by itself disturbing effects.

2. The magnitude of this perturbation appeared to be sufficiently constant and close to one hundred gammas; it is not in a direct relation with either the character of the diked terrain or the magnitude of faulting.

3. In the cases considered in this paper this perturbation is sufficiently considerable to conceal the theoretic anomaly owing to the difference in the magnetic susceptibility of the terrains in contact.--W. Ayvazoglou.

(703) CHARACTERISTICS OF THE DASHKESAN IRON-ORE DEPOSIT BASED ON DATA OBTAINED FROM A MAGNETOMETRIC SURVEY CARRIED OUT IN 1923 AND 1924 (IN RUSSIAN)

By D. Ortenberg

Transactions of the Geological and Prospecting Service of the U.S.S.R., Leningrad, No. 11, 1930, 38 pp.

This article is divided into three parts: (1) Geological, (2) data obtained from the magnetometric survey, and (3) considerations on the calculation of the ore reserves. The magnetometric survey was carried out with

Tiberg-Thalen's magnetometers. Six and one-half square kilometers were surveyed, and there were 2,400 stations. The purpose of the survey consisted in establishing the contours of the deposit and in verifying them, by geological observations. The results of the investigation are given in a series of maps.

Based on the extent of the ore-bearing areas outlined by the magnetometer survey and taking the average thickness of the ore-bearing stratum of the garnet-magnetite rocks of the northern group as attaining 25 meters, an ore reserve of about 17 million tons is calculated.--W. Ayvazoglou.

(704) A NEW MAGNETIC ANOMALY IS DISCOVERED (IN RUSSIAN)

By A. Strona

Vestnik of the Geological and Prospecting Service, U. S. S. R.,
Leningrad, vol. 6, No. 1-2, 1931, p. 60.

Based on magnetometric survey carried out with Tiberg-Thalen's magnetometer a very interesting anomaly was established in the Rognedin region (about 100 km. from Briansk and 350 km. from Moscow). The disturbing ore deposits are expected to be at a shallow depth of from 100 to 200 meters.--W. Ayvazoglou.

(705) MAGNETISCHE UNTERSUCHUNGEN BEI BERGGIESSHÜBEL IN SACHSEN

(MAGNETIC INVESTIGATIONS NEAR BERGGIESSHÜBEL IN SAXONY)

By Gerhard Neumann

Gerlands Beiträge zur Geophysik, Ergänzungshefte für angewandte Geophysik,
Leipzig, vol. 2, No. 1, 1931, pp. 22-68.

Contents of the article:

A. Introduction: Reason for Investigation.
B. Chapter I: Boundaries, geographic position, and geology of the region under investigation.
   Chapter II: Carrying out of measurements: (1) Plan of measurements (leading idea of the investigation; single parts of the investigation; distribution of stations); (2) instruments (determination of the constants; reduction of field measurements).
   Chapter III: Physicogeological part: (1) Distribution of the vertical intensity and relations to the underground (general remarks; magnetic rocks inside of the contact area; disturbances above the ore layers and their continuation; continuations of the whole disturbance toward the northwest; comparative measurements in the environs); (2) comparative investigations of the rock magnetism in the region of investigation (method; results).

Chapter IV: On the interpretation of the Martinzecher anomaly: (1) General remarks; (2) special investigation (considering of cases possible theoretically; practical results of measurements below the ground; calculations); (3) conclusions.

C. Conclusion: Summary of the results.
Literature.

The author's abstract reads as follows:

Magnetic measurements were made in the environs of Berggiesshübel (Saxony). A very extensive positive anomaly was found north of the negative zone, which is known by previous investigations above the main deposit of magnetic iron. It is shown that the positive anomaly is due to magnetite-bearing igneous-metamorphic rocks on top of the granite massif of Markersbach. Samples of magnetic material were tested. In addition to vertical intensity, measurements of horizontal intensity were made. A subsurface investigation shows the polarization of the magnetic zone which is irregular. The large body of magnetite-bearing rocks has probably induced the opposite magnetization in the iron deposit.

Ten figures complete the article.--W. Ayvazoglou.

(706) DIE VERMESSUNG DER ERDMAGNETISCHEN ANOMALIE BEI PR.-EYLAU IN OSTPREUSSEN UND EIN VERSUCH IHRER DEUTUNG

(MEASUREMENT OF THE EARTHMAGNETIC ANOMALY NEAR PR.-EYLAU IN EASTERN PRUSSIA AND AN ATTEMPT AT ITS INTERPRETATION)

By O. Baseler

Gerlands Beiträge zur Geophysik, Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 2, No. 1, 1931, pp. 69-121.

Earthmagnetic measurements with three Schmidt field balances were made by the author in 1929 and 1930 near Preussisch-Eylau in East Prussia over a region of about 50 by 50 km.^{2} Z was measured at 309 stations, and H at 124 stations; D was already known. The first part of the article deals with the proper measurements. The accuracy of adjustment of the three instruments used, as well as corrections for temperature and diurnal variations are briefly discussed; the determination of scale values is examined (1p = C 28 γ for Z, 1p = C 15 γ for H). The mean error of single measurements was from 15 to 20 γ. For the calculation of the normal field Nippoldt's formula for East Prussia was used. The system was compared with figures obtained in the North German Survey, for Nippoldt's Wickbold anomaly and for Gross-Raum.

The second part deals with the reasons for the anomaly and with attempts to determine the depths. The disturbances can not be caused either by the diluvium or by the mesozoic or paleozoic layers, thus crystalline subsoil must be the reason. An attempt to determine the depth was made, based on a sphere and a suitable ellipsoid according to Koenigsberger, with the aid of the intersection point of the disturbance vectors, by the comparison with profiles of cylinders and a level layer; from 3 to 4 km. were established as the most probably value for the nearest depth of the disturbing mass; the value of the susceptibility was found to be equal to 0.05; the uncertainty of all the values is, of course, pointed out by the author. Comparisons with some other anomalies are also made and Haalck's diagram method is discussed.--Author's abstract translated by W. Ayvazoglou.

(707) MAGNETOMETER SURVEY OF JACKSON AREA

By L. Spraragen

The Oil and Gas Journal, Tulsa, vol. 30, No. 36, 1932, pp. 14-15 and 83-84.

Considerable publicity on the possibility of obtaining oil and gas in the Jackson area, Mississippi, was given by Oliver Hopkin's report published by the U. S. Geological Survey in Bulletin 641. A few paragraphs from this report, dealing with the surface structure, are mentioned in this article. The author was called upon to make several small tract surveys in the vicinity of Jackson with the view of determining favorable drilling locations. For this reason the magnetometer survey shows that the readings are more or less grouped in small areas instead of being uniformly distributed throughout the area. The results of the survey are given in a map. Comparing the surface structure map with the magnetometer survey it can be observed that the use of the magnetometer has given additional information with respect to the top of the structure, supplementing the work of the surface geologist.

Comparing the magnetometer survey with the subsurface map it was established that the high (shown in section 15 of the map) indicated with the magnetometer checked closely with the high outlined by the actual drilling.

The comparison of a cross section of the Jackson structure as indicated by the magnetometer results and the actual drilling operations is given in a map.--W. Ayvazoglou.

3. SEISMIC METHODS

(708) SOME NEAR EARTHQUAKES REPORTED IN THE "INTERNATIONAL SEISMOLOGICAL SUMMARY"

By R. Stoneley

Monthly Notices of the Royal Astronomical Society, London, Geophysical Supplement, vol. 2, No. 7, 1931, pp. 349-362.

The problem raised by the author in this article consists of discovering what use can be made at present of the data supplied by the separate stations

and published in the International Seismological Summary, and whether these large residuals are explainable.

The following are the headings of the article:

Method and data.
Identification of the different waves.
Improvement of epicenter.
Least square determinations of velocities.
Evidence for a wave of velocity about 7.0 km/sec.

In conclusion Stoneley expresses the hope that Professor Turner's plan of printing <u>all</u> the additional readings will be continued, as their value is clearly indicated by the present investigation.--W. Ayvazoglou.

(709) THE THICKNESS OF THE CONTINENTAL LAYERS OF EUROPE

By Robert Stoneley

Monthly Notices of the Royal Astronomical Society, London, Geophysical Supplement, vol. 2, No. 8, 1931, pp. 429-433.

Data concerning the periods and group-velocities of Love waves are found by measurement of selected seismograms.

To separate the Love waves from Rayleigh waves, records are chosen in which the waves reach the recording station in an easterly azimuth.

The waves of group-velocities greater than 3.7 km/sec. give 12 km. for the thickness of the granitic layer. If velocities down to 3.5 km/sec. are included, the corresponding thickness of the granitic layer is 13 km.

The inclusion of lower group-velocities would require taking the sedimentary layer into account. The measures afford data for studying this effect, but the formulas become much more complicated.--Author's abstract.

(710) THE TIMES OF P AND S AT SHORT EPICENTRAL DISTANCES

By Harold Jeffreys

Monthly Notices of the Royal Astronomical Society, London, Geophysical Supplement, vol. 2, No. 8, 1931, pp. 399-407.

A recent discussion (The revision of seismological tables; see Geophys. Abs. 32, p. 313) of the transmission times of closely observed earthquakes, recorded in the International Seismological Summary, has led to a set of corrections to the existing tables, which appear satisfactory at distances up to about 10° and over 25°, but which present some uncertainties in the intermediate range. It was hoped that the causes of these would be traced by means of a narrower classification, which is carried out in the present paper.

The author's summary reads as follows:

1. Examination of the data contained in the International Seismological Summary, and of Byerly's results for the Montana earthquake, leads to the conclusion that for $\Delta < 25^\circ$ the times of transmission for P and S, apart from constant terms, closely fit the formulas

$$T_P = 14.30\,\Delta - 2.00\,(\Delta/10)^3$$
$$T_S = 25.70\,\Delta - 3.50\,(\Delta/10)^3$$

The cube terms are about double those given previously.

2. P and S at stations within this range are usually followed by other pulses at intervals of about 8s. It seems probable that the curious behavior of the S residuals, derived from the "Summary," is due to the reading of one or other of these later pulses for S.

3. The later pulses may be due to internal reflection in the upper layers.--W. Ayvazoglou.

(711) ON THE CAUSE OF OSCILLATORY MOVEMENT IN SEISMOGRAMS

By Harold Jeffreys

Monthly Notices of the Royal Astronomical Society, London, Geophysical Supplement, vol. 2, No. 8, 1931, pp. 407-416.

At a distant point the disturbance takes the form of three movements of P, S, and Rayleigh type, respectively, each consisting of a single displacement and return; there is no sign of a train of oscillations.

In an actual seismogram the motion is oscillatory throughout. From the arrival of P to the dying away of the coda there is a continual vibration, the later phases being recognizable only by increases in the amount of the movement of changes in its character. For the surface waves this feature is adequately explained by dispersion due to the presence of several distinct surface layers. But this cause does not contribute to the vibrations that occur before any surface waves have arrived.

This paper discusses the question of whether dispersion in the bodily waves can be of such a character as to explain this motion.

In this article Jeffreys discusses this problem in connection with the heterogeneity of the earth, gravitation between its parts, and possibility of internal friction.

In the paragraph "Reflection in the surface layers" the author says that his investigations exhaust the possibilities of accounting for the oscillations between P and S, or between S and L, by any kind of dispersion in the body of the earth. All the factors considered lead only to results quantitatively inadequate. The only suggestion that survives is that the oscillations are due to reflections of the original pulse within the surface layers and are refracted partially into the lower layer at each arrival there.

In the last paragraph of the article Jeffreys gives a mathematical treatise on the spherical distortional pulse.--W. Ayvazoglou.

(712) THE REFLECTED WAVES FROM DEEP FOCUS EARTHQUAKES

By F. J. Scrase

Proceedings of the Royal Society, London, Series A, vol. 132
No. A 819, pp. 213-235.

The effect of an abnormally deep focus on the reflected waves of earthquakes is considered. In general a number of supplementary reflected waves may occur and if the focus is sufficiently deep, they should produce definite separate phases on the records. The times of travel of both the supplementary waves and the more normal waves have been derived for several depths of focus, C. G. Knott's paths of longitudinal and transverse waves being taken as a basis.

It is found that the commencements of the additional phases can generally be recognized on the seismograms and that the times of transit are in reasonable agreement with the calculated times. This, it is considered, is definite confirmation of the occurrence of deep focus earthquakes. Further, the appearance of the supplementary reflected waves provides a means of recognizing a deep focus earthquake from the records of a single station.

The results of the investigation favor the idea that the initial phase of an earthquake is a direct compressional wave and is not generated by reflection of a distortional wave.--Author's abstract.

(713) SOME EXPERIENCE IN SEISMIC PROSPECTING

By V. Gavrilovich Gabriel

Gerlands Beiträge zur Geophysik, Ergänzungshefte für Angewandte Geophysik, Leipzig, vol. 2, No. 1, 1931, pp. 122-130.

The author discusses some observations connected with seismic exploration in practice and the conclusions which can be drawn from them. The seismic trial party was made up of five seismographs of Schweidar Mintrop mechanical type with five radio receiving and transmitting sets. The results obtained are reported as follows:

The study of the values obtained show that up to 100 meters the results from the dropping of a 200-pound mass of lead are more or less comparable in speed and intensity with these from explosives. Also the results show that with a few exceptions there is only a slight deviation in the values of the distances obtained by surveying and by computing from the "Sound Velocity in Air" charts. Unfortunately the difficulties in handling the heavy masses during the field work prohibit, under the present conditions, an extensive use of heavy weights in seismic prospecting. Perhaps the dropping of heavy blocks might be justified in exploration concerned with shallow seated layers or in seismic training parties connected with the universities and geophysical research laboratories.

Tables are added giving data obtained from (1) explosion, (2) dropping of heavy weights, and (3) profile shooting over anticline.--W. Ayvazoglou.

(714) WAVE EQUATION FOR THE CASE OF A HETEROGENEOUS MEDIUM (IN RUSSIAN)

By S. Sobolev

Académie des Sciences de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 6, 1930, pp. 1-57.

A mathematical discussion relating to the solution of the wave equation in the case of a heterogeneous medium is given.--W. Ayvazoglou.

(715) CONCERNING THE WAVE EQUATION FOR THE CASE OF A HETEROGENEOUS ISOTROPIC MEDIUM (IN FRENCH)

By S. Sobolev

Académie des Sciences de l'U.R.S.S., Publications of the Seismological Institute, Leningran, No. 2, 1930, pp. 163-167.

A mathematical discussion concerning the wave equation for the special case of a heterogeneous isotropic medium is given.--W. Ayvazoglou.

(716) EPICENTRAL ZONE OF CRIMEAN EARTHQUAKES (IN RUSSIAN)

By N. V. Rayko

Académie des Sciences de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 3, 1930, pp. 1-13.

After the disastrous earthquakes in Crimea in 1927 the Seismological Institute established a series of stations for studying earthquakes occurring at small distances from them. Observations were made in Feodosia, Yalta, Simferopol, and Sevastopol. The stations were provided with Nikiforov's horizontal seismographs with optical registration. Two hundred and sixteen earthquakes were registered during the period from March 13, 1928, to October 1, 1929. The data obtained are given in tables. A map showing the geographic distribution of the epicenters is added.--W. Ayvazoglou.

(717) SEISMOMETRIC INVESTIGATION OF SEVERAL BRIDGES IN LENINGRAD (IN RUSSIAN)

By N. V. Veshniakov

Académie des Sciences de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 4, 1930, pp. 1-20.

The following three bridges were investigated: The Lieutenant Schmidt bridge and the Liteyny bridge across the Neva, and the Siniy bridge across the Moyka.

Wiechert-Mintrop's, Nikiforov's, and Galitzin's seismographs were used. The results of investigations are shown in diagrams and tables.--W. Ayvazoglou.

(718) INVESTIGATION OF EQUILIBRIUM CONDITIONS OF EARTHERN MASSES UNDER THE ACTION OF SEISMIC FORCES (IN RUSSIAN)

By V. Tsshokher

Académie des Sciences de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 5, 1930, pp. 1-11.

In this article the author discusses the action of the seismic accelerations upon the equilibrium conditions of earth masses, such as embankments, excavations, etc. Embankments made from sand, loess, and cement were made on a special platform to which accelerations of from 100 to 3,500 mm/sec.^{2} were communicated.

The author derives a theoretic formula which gives a sufficiently correct estimate of the character of the phenomenon.--W. Ayvazoglou.

(719) ON THE DIFFRACTION OF SPHERICAL ELASTIC WAVES NEAR THE SURFACE OF A SPHERE (IN RUSSIAN)

By S. Sobolev

Académie des Sciences de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 7, 1930, pp. 1-13.

In this article the author gives a mathematical discussion of the diffraction of spherical elastic wave near the surface of a sphere for the case of longitudinal waves only.--W. Ayvazoglou.

(720) PLAN QUINQUENNAL DES TRAVAUX DE RECHERCHE SCIENTIFIQUE DE L'INSTITUT SÉISMOLOGIQUE DE l'ACADÉMIE DES SCIENCES DE l'U. R. S. S.

(FIVE-YEAR PLAN OF SCIENTIFIC RESEARCH WORK IN THE SEISMOLOGICAL INSTITUTE OF THE ACADEMY OF SCIENCES OF THE U.S.S.R.)

By P. M. Nikiforov

Académie des Science de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 9, 1930, pp. 1-27.

In the preface of his book Nikiforov expresses the hope that all scientific institutions similar to the Seismological Institute of the Academy of Sciences in Leningrad, will be interested in the 5-year scheme of seismological research presented and will give critical attention to the subject of this plan, as well as to considerations by which the plan can be made more complete.

The plan is to be carried into effect from October 1, 1929 to September 30, 1933.

The first part of the book deals with the (1) principal purposes of the Seismological Institute, (2) structure of the Seismological Institute, and (3) distribution of seismic stations.

In the second part the thematic plan of the scientific research work of the Seismological Institute is discussed.

The article is concluded by the financial plan. A map of the U.S.S.R. showing the distribution of the first-class stations, and the regional stations over the whole country is added.--W. Ayvazoglou.

(721) ON THE PROPAGATION OF ELASTIC WAVES ALONG THE SURFACE OF SEPARATION OF TWO MEDIA HAVING DIFFERENT ELASTIC PROPERTIES (IN RUSSIAN)

By V. Kupradze and S. Sobolev

Académie des Sceiences de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 10, 1930, pp. 1-23.

The translation of authors' abstract reads as follows: In his work, On the Propagation of Tremors Over the Surface of Elastic Solid, Sir H. Lamb considered the propagation of the elastic waves along the surface of an elastic space separated by vacuum. The authors of this article studied, based on Lamb's method, the propagation of elastic waves along the surface separating two media having different elastic properties. Instead of the vacuum a compressible liquid is supposed in which the velocity of the longitudinal waves is smaller than the velocity of the Rayleigh waves in the second medium. The character of the propagation of the elastic vibrations is essentially changed owing to the presence of the second medium.

The combined influence of the two media upon the displacement of 1 molecule of the surface of the elastic body produces a new maximum, the properties of which are similar to the Rayleigh waves and the velocity of which is a little smaller than the velocity of the longitudinal waves of the liquid.

In case of a liquid of small density the influence becomes negligible.--W. Ayvazoglou.

(722) ON A LIMITED PROBLEM OF THE THEORY OF THE LOGARITHMIC POTENTIAL AND ITS APPLICATION TO THE REFLECTION OF THE PLANE ELASTIC WAVES (IN RUSSIAN)

By S. Sobolev

Académie des Sciences de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 11, 1930, pp. 1-18.

The author gives the solution of the following problem: To determine a function v, harmonic inside of a certain domain and satisfying along the contour 1 of this domain the following condition:

$$p(s)\frac{\partial v}{\partial n} + 9(s)\frac{\partial v}{\partial s} = f(s)$$

in which s is the length of the arc of 1 and n the direction of the normal to 1.

This problem may be applied to the investigation of the reflection of transversal waves for the case in which the angle of incidence is greater than the critical angle of total reflection.--Author's abstract translated by W. Ayvazoglou.

(723) ON THE POSSIBILITY OF OBSERVING A MOHOROVICIC PHASE IN THE SEISMOGRAMS OF EARTHQUAKES IN THE CAUCASUS (IN RUSSIAN)

By N. W. Rayko

Académie des Sciences de l'U.R.S.S., Publications of the Seismological Institute, Leningrad, No. 12, 1930, pp. 1-10.

Mohorovicic phases represented in Tables 1 and 2 of this article by the symbol P_M have been revealed by the two Leninakan earthquakes of October 22, 1926 (seismograms of the first-class station at Baku and second-class station at Piatigorsk). That this phase P_M really represents the Mohorovicic phase is confirmed by the computation of the propagation velocity of the wave P_M recorded in Table 4 (the epicenters and the moments of the beginning of the Leninakan earthquakes were computed according to the data obtained from the stations Pulkovo, Sverdlovsk, and Tashkent).

According to Table 4, the mean value of the velocity of V_{P_M} proved to be equal to 5.75 km/sec., whereas the foreign authors have found somewhat smaller values for the velocity of propagation of seismic waves in the surface layer of Central Europe. This may be explained by the fact that the surface layer in the Caucasus is composed of more elastic rocks than those in Central Europe.

The Mohorovicic phase was also observed during the Kuban earthquake of April 19, 1926, at the station of Piatigorsk and at the station of Makeevka.--Author's abstract.

4. ELECTRICAL METHODS

(724) GEOPHYSICAL PROSPECTING. RECENT DEVELOPMENTS IN ELECTRICAL INSTRUMENTS

Editorial note

Industrial Australian and Mining Standard, Melbourne, vol. 86, No. 2220, 1931, p. 376.

This is an editorial report on a paper read by A. S. Eve before the British Association for the Advancement of Science in London on September 25, 1931, in which he presented a comparative study of the three different electrical methods of prospecting and gave a brief description of the methods employed.

The three methods described were the self-potential, the Radiore, and the Gish-Rooney, as well as Lee's modification of the latter.--W. Ayvazoglou.

(725) DISCUSSION ON "THE METHOD OF THE GROUND RESISTIVITY MAP AND ITS PRACTICAL APPLICATIONS"

By C. and M. Schlumberger

The Canadian Mining and Metallurgical Bulletin, Montreal, 1931, No. 236, pp. 1413-1414.

This is a brief discussion on the article published by C. and M. Schlumberger under the same title in the Canadian Mining and Metallurgical Bulletin of February, 1931, No. 226, pp. 271-294 (see Geophys. Abs. 23, p. 73). --W. Ayvazoglou.

(726) ELECTRIC DIVINING ROD OPENS RICH GOLD MINE

Editorial note

Washington News, February 12, 1932.

According to information received from Skelleftea (Sweden), a valuable gold-bearing ore has been located near Boliden, thanks to a new electrical divining rod. A village has been built at the mine and a smelter set up.

From 600,000 tons of ore the smelter will extract each year from 10 to 12 tons of gold, twice that amount of silver, and several thousand tons of copper, sulphur, and arsenic besides.

A description of the principle of the new divining rod is not given.--W. Ayvazoglou.

5. RADIOACTIVE METHODS

(727) LA PROSPECCION GEOFISICA POR EL METODO RADIO-ACTIVO

(GEOPHYSICAL PROSPECTION BY RADIOACTIVE METHOD)

By D. Shakhnazarov

Petroleos y Minas, Buenos Aires, vol. 11, No. 128, 1931, pp. 7-9.

In the beginning of this article the author mentions briefly the principles of radioactive exploration, the approximate amount of radium extracted so far in various regions of the earth, and the price of it. The methods of radioactive exploration for minerals are described in the second part of the article.

In conclusion Shakhnazarov mentions the electroscopic method established by Pierre Curie and the apparatus constructed, by the application of which good results may be obtained in geophysical prospecting.--W. Ayvazoglou.

(728) ON THE DEPOSITS OF THE RADIOACTIVE MINERAL FORMATIONS IN THE DISTRICT OF KHAKASSK (FORMERLY PROVINCE OF ENISSEIYSK) (IN RUSSIAN)

By G. Labasine

Transactions of the Geological and Prospecting Service of the U.S.S.R., Leningrad, No. 19, 1930, 56 pp.

By prospecting work the author established radioactive substances in the following three formations in the Khakassk region: (1) In the pegmatites and aplites of the deep rocks; (2) in the secondary, mainly colloidal, minerals, and (3) in rocks containing rich carboniferous residues.

In the samples taken from the region of the Boulan-Koul Lake, the Tyrdov massif, the cupriferous sandstones of Tchagys-Karygy, the carboniferous rocks of the Askyz-Abakane basin and the colloids of the Kluichevaya mountain the ratio of U to Th varied between 4.4 and 43.9. Especially marked prevalence of thorium was established in the pegmatites of the Tyrdanov massif (Th : U = 42.7) and in the colloids of the Kliuchevaya mountain (Th : U = 43.9).

The region is considered to be of great scientific interest.--W. Ayvazoglou.

(729) RADIOACTIVITY AND THEORIZING

By Bailey Willis

American Journal of Science, New Haven, vol. 23, No. 135, 1932, pp. 193-226.

Since 1915 several geologists and geophysicists have framed hypotheses regarding the distribution of radioactive minerals in the earth and the heat effects attributed to them. Holmes, Joly, Chamberlin, Barrel, and L. H. Adams have published more or less elaborate discussions.

In this article the several hypotheses are considered from the point of view of the method of multiple hypotheses, each of them being briefed and subjected to the test of the verity of its fundamental assumptions. The task is simplified by grouping the theories under two heads, namely those that postulate some regular distribution of radioactive minerals in the globe versus those that assume irregularity of distribution. Reasons are assigned for regarding the former as inconsistent with the facts and consequently unreal, whereas the latter group of discussions appears to contain the seed of fruitful progress toward an understanding of the thermal, eruptive history of the earth.--Author's abstract.

7. UNCLASSIFIED METHODS

(730) TRABAJOS DEL LABORATORIO GEOFISICO

(WORK CARRIED OUT IN THE GEOPHYSICAL LABORATORY)

By D. Shakhnazarov

Petroleos y Minas, Buenos Aires, vol. 11, No. 124, 1931, pp. 4-8.

The purpose of this article consists in giving a summarized description of methods of geophysical prospection. The following items are discussed:

1. Prospecting for minerals by the magnetic method; historical review.
2. Principles of magnetometric exploration for minerals.
3. Determination of magnetic properties of minerals and rocks.
4. Determination of gravity of minerals and rocks.
5. Determination of radioactivity of minerals and rocks.
6. Collection of samples.

Maps showing geophysical properties of minerals as established from the data obtained in geophysical laboratories will be of great use in geophysical prospecting.--W. Ayvazoglou.

(731) ESTUDIOS GEOFISICOS. UN FACTOR DE ACTUALIDAD EN EL DESARROLLO DE LA MINIERA

(GEOPHYSICAL STUDIES. A FACTOR OF PRESENT IMPORTANCE IN THE DEVELOPMENT OF MINING)

By D. Shakhnasarov

Petroleos y Minas, Buenos Aires, vol. 11, No. 126, 1931, pp. 5-9.

The importance of geophysical prospection (magnetic, gravimetric, radioactive, electrical and seismic methods) is emphasized. A brief description of the development of geophysics in various countries (Germany, United States, Spain, France, Hungary, Mexico, Russia, and Yugoslavia) is given. The costs of geophysical exploration by different methods are discussed.--W. Ayvazoglou.

(732) BRIEF SUMMARY OF THE WORK CARRIED OUT IN 1930 IN THE SCIENTIFIC RESEARCH DEPARTMENT OF THE GEOLOGICAL AND PROSPECTING SERVICE OF THE GEOPHYSICAL INSTITUTE (IN RUSSIAN)

By V. Andreev and V. Chernobrovin

Vestnik of the Geological and Prospecting Service in the U.S.S.R., Leningrad, vol. 6, No. 1-2, 1931, pp. 43-47.

The Geophysical Institute carried on investigations along the following six branches:

1. Electrical. Development of electrical methods of prospecting, improvement in construction of apparatus, training of personnel.
2. Magnetic. Comparative measurements for the determination of the accuracy of the field instruments. The system of suspension of the magnets of the coil of a deflector-magnetometer was improved by applying bifilar suspension.
3. Gravitational. Magnetic properties of materials used for manufacturing variometer beams were investigated; beams and threads manufactured in Russia were tested; Vening-Meinesz' pendulum apparatus was investigated; the temperature of the constants of Stückrat's pendulum apparatus was investigated.
4. Seismic. The theory of the seismic method of prospecting was developed.
5. Radioactive. Laboratory tests on various branches of the radioactive method of prospecting were carried on. The radioactivity of samples of more than 5,000 boxes of geological collections was investigated.
6. Geothermal and stratigraphic. Apparatus used for geothermal investigations was tested; a new electrical thermometer for depths to 750 meters was constructed. A new apparatus for measuring crooked boreholes was prepared.

Field operations were carried out on a larger scale; the number of field parties increased from 43 in the previous year to 80; the area surveyed increased from 2,757 km.^{2} to 7,995 km.^{2}--W. Ayvazoglou.

(733) GEOPHYSICAL SOCIETY MEETS AT NEW ORLEANS

Editorial note

The Oil and Gas Journal, Tulsa, vol. 30, No. 33, 1932, p. 111.

The meeting of the Society of Petroleum Geophysicists, held at New Orleans, was opened on December 30, 1931. The following 13 technical papers have been scheduled for the program:

(1) Eötvös torsion balance, by Donald C. Barton; (2) A new instrument for measuring very small differences in gravity, by Kenneth Hartley; (3) Charts for torsion balance readings, by M. M. Slotnick; (4) The calculation of the motion of the ground for seismograph records, by H. A. Wilson; (5) Amplitude relations in seismic prospecting, by W. Maurice Ewing; (6) Asymetry of sound velocity in stratified geologic formations, by Burton McCollum and F. A. Snell; (7) Velocity of elastic waves in granite, by L. Don Leet and W. Maurice Ewing; (8) The reflection seismograph -- an application, by Eugene McDermott; (9) Seismological discovery and partial detail of the Vermillion Bay Salt Dome, by E. E. Rosaire and O. C. Lester, jr.; (10) Some special cases of the reflection and refraction of seismic waves between similar rocks with application to the study of crustal layers by distant quakes, by Louis B. Slichter and V. Gabrilovich Gabriel; (11) The correlation of isogeothermal surfaces with the rock strata, by C. E. Van Orstrand; (12) Geothermal gradient determinations in the Lake Superior copper mines, by L. R. Ingersoll; (13) Electrical prospecting as applied in locating oil structures, by Leo J. Peters and John Bardeen.--W. Ayvazoglou.

8. GEOLOGY

(734) ASYMMETRIC FOLDS WITH REFERENCE TO GERMAN SALT BODIES

By Hans Stille

Bulletin of the American Association of Petroleum Geologists, Houston, vol. 16, No. 2, 1932, pp. 169-177.

A brief discussion is presented of the conditions of asymmetry and overhang in the salt domes of Germany, Reference is made to the folded region of central Spain, and the orogeny here is likened to that in Germany. This paper is offered to facilitate the comparison between German and American salt-dome phenomena.--Author's abstract.

9. NEW BOOKS

(735) Freeman, John Ripley. Earthquake damage and earthquake insurance. McGraw-Hill Book Co. (Inc.), New York and London, 1932, 904 pp. Price, $7. The book contains studies of a rational basis for earthquake insurance and of engineering data for earthquake-resisting construction. Table of contents: (1) The present situation and the scope of the present study; (2) Earthquake motion and causes of earthquakes; (3) The ordinary measure of earthquake violence; (4) Frequency and violence of earthquakes in various parts of the United States and Canada; (5) The narrow zone of destruction by earthquakes; (6) Structural lessons and loss ratios from the San Francisco and Charleston earthquakes; (7) Relation of structural safety to the local stability of the ground; (8) Structural lessons and loss ratios from other important American earthquakes; (9) Lessons about earthquake-resisting buildings from the Japanese earthquake of 1923; (10) Lessons from Italian earthquakes; (11) Other interesting earthquake lessons from Nicaragua and New Zealand; (12) Prediction of time, place of occurrence and damage of future earthquakes; (13) Some tentative figures on probable average earthquake loss ratio for various localities and various types of buildings; (14) Recent rates of premium for earthquake insurance by stock insurance companies; (15) Affiliation of earthquake insurance with fire insurance; (16) Municipal building codes for protection against earthquake damage; (17) Textbooks on earthquake-resisting structural design; (18) Researches with shaking tables; (19) The motion of the ground in an earthquake; (20) The design of earthquake-resisting buildings; (21) Data from seismograms, etc.; (22) Suggestions for a program of earthquake research.--W. Ayvazoglou.

(736) Haddock, M. H. Deep borehole surveys and problems. McGraw-Hill Book Co. (Inc.), New York, N.Y., 1932, 296 pp., 186 figs., $4. The important and accepted means of surveying the courses of deep boreholes and orientating their cores are explained. The book traces the evolution of modern borehole-surveying devices, and adds various problems relevant to strata, location, and orientation. Chapter headings: (1) Deviation and its causes; (2) Auxiliary registrations in borehole surveys; (3) Instrumental survey of boreholes; (4) Fluid methods of surveying boreholes; (5) Core orientation; (6) Compass and plumb-bob methods; (7) Pendulum methods; (8) Photographic methods; (9) Gyroscopic compass methods of surveying boreholes; (10) Geophysical methods of investigating boreholes; (11) Problems; (12) Bibliography.--W. Ayvazoglou.

(737) Lahee, Frederick H. Field geology. McGraw-Hill Book Co. (Inc.), New York and London, 3d ed., 1932, 789 pp., 538 figs., $5. Pocket size, flexible. This book is intended to serve as both textbook and pocket manual on those phases of geology involved in field work. The book contains a chapter dealing with geophysical surveying, including the torsion balance, magnetic, seismic, and electrical methods.--W. Ayvazoglou.

10. PATENTS

(738) ELECTROMAGNETIC WAVE EXPLORER

Ralph W. Deardorff, of Contra Costa County, California

U. S. Patent 1,838,371. Patent issued December 29, 1931.

This invention relates to the locating of mineral deposits, such as oil or ores, by the aid of directional transmission and detection of electromagnetic radiations.

The objects of the invention are: To make it possible to determine the character of the deposits by causing them to act as reflectors of electromagnetic radiant energy, and by ascertaining the quality of the reflection as a function of the frequency used for the radiations; and to make it possible to investigate the character of the invisible deposits by determining the speed at which the radiations pass through the medium formed by the deposits.

Claims allowed - 3.

(739) ELECTRICAL PROSPECTING

Charles R. Nichols and Samuel H. Williston, of Dallas, Texas

U. S. Patent 1,841,376. Patent issued January 19, 1932.

The present invention is characterized by the creation of such an electric current flow through the earth as to impress an identifiable potential condition or characteristic on a portion of the earth's surface, and by the determination of the direction and extent of the displacement, if any, of such portion from a geographical location which is fixed by the arrangements creating such current flow, and with which said surface portion will or will not coincide, accordingly as the distribution of the current flow through the earth adjacent to said location is not or is significantly affected by an earth body or bodies of different conductivity from the adjacent earth.

Claims allowed - 23.

(740) ELECTRICAL PROSPECTING

Charles R. Nichols and Samuel H. Williston, of Dallas, Texas.

U. S. Patent 1,841,975. Patent issued January 19, 1932.

The present invention is characterized by the provisions made for the effective use of the portion of the energizing circuit or circuit system which is external to the earth itself, to establish a definite or base potential at a point in the said circuit system which may be utilized in determining the potential at the exploration point, or in making measurements dependent on the difference between the said base potential and the exploring electrode potential.

The present invention is especially characterized by the manner in which the base potential point in the energizing circuit is established, and in particular by the steps taken to compensate for the variable and ordinarily unknown, contact resistance between the earth and the external energization circuit at each point of energization.

Claims allowed - 8.

(741) ELECTRICAL PROSPECTING

Charles Rodney Nichols and Samuel H. Williston, of Dallas, Texas.

U. S. Patent 1,841,976. Patent issued January 19, 1932.

The present invention relates to methods of and apparatus for determining the location and character of subsurface bodies of earth portions differing in electrical resistance from adjacent earth portions by creating potential differences between separated portions of the earth surface to induce current flows through the earth underlying the exploration field.

The invention provides a method and apparatus adapted to rapid reconnaissance work along a line extending across the field to be explored, as well as means for making effective use of the circuit system to establish a base potential which may be utilized in determining the potential at an exploration point.

Claims allowed - 6.

(742) ELECTRICAL PROSPECTING

Ralph M. Nichols, of Dallas, Texas, assignor of one-half to Charles R. Nichols and one-half to Samuel H. Williston, both of Dallas, Texas.

U. S. Patent 1,841,977. Patent issued January 19, 1932.

The general object of the present invention is to provide an improvement, characterized by the simplicity of apparatus and operations required, in the method of determining the location and character of subsurface bodies or earth portions of different electrical resistance from the adjacent earth portions, by creating an electric current flow through the earth and measuring electrical effects produced at the surface of the earth resulting from such current flow.

Claims allowed - 3.

(743) ELECTRICAL PROSPECTING

Charles R. Nichols, of New York, N. Y., and Samuel H. Williston, of New Haven, Connecticut.

U. S. Patent 1,842,361. Patent issued January 19, 1932.

The general object of this invention is to provide improved methods for detecting the location and character of subsurface bodies or earth portions of different electrical resistance from the adjacent earth portions.

The method consists in energizing distributed points in a field of observation by means of a source of alternating current so as to create isopotential lines which intersect at a point, and in locating the said point by means of a detector circuit including an antenna movable over the exploration field.

Claims allowed - 1.

(744) ELECTRICAL PROSPECTING

Charles R. Nichols and Samuel H. Williston, of Dallas, Texas.

U. S. Patent 1,842,362. Patent issued January 19, 1932.

The method described in the present invention consists in energizing an observation field by causing an electric current flow through the earth between points thereof on which different potentials are impressed, as by means of electrodes inserted in the earth and connected to the terminals of an electric generator or generators, in order that the resultant current flow distribution through the earth may be determined, or compared with the current distribution which would result from such energization if the earth beneath the observation field were of uniform conductivity.

Claims allowed - 11.

(745) SEISMOGRAPH

Sepp Horvath, of Houston, Texas.

U. S. Patent 1,842,968. Patent issued January 26, 1932.

This investigation relates to an improvement in seismographs and recording mechanisms employed in locating geological structures. The invention involves the use of instruments embodying the electrical transfer of mechanical vibrations of the ground into corresponding electrical alterations of conditions of electrical circuits. With electrical circuits of the type here used very sensitive measurements of these mechanical vibrations can be had by means of high-frequency circuits.

Claims allowed - 9.

(746) TORSION BALANCE AND THE LIKE

Herman Shaw, of London, and Ernest Lancaster Jones, of Berkshire, England, assignors to Geophysical Research Corporation, of New York, N. Y., a corporation of New Jersey.

U. S. Patent 1,843,342. Patent issued February 2, 1932.

This invention relates more particularly to improvements in the design and arrangement of torsion balances of the type in which one portion of the suspended system is maintained at a definite vertical distance above or below other portions of that system, as, for example, in balances of the Eötvös and similar types.

It further relates to improved arrangements in the distribution of the essential parts of any such suspended system, and to the improved arrangement and simultaneous operation of one or more such systems forming a complete instrument. The object of the invention is to provide means for obtaining fewer settings of the instrument than has hitherto been possible.

Claims allowed - 7.

(747) DETERMINATION OF SUBSURFACE FORMATIONS

John Clarence Karcher, of Montclair, N. J., assignor to Geophysical Research Corporation of New York, N. Y., a corporation of New Jersey.

U. S. Patent 1,843,725. Patent issued February 2, 1932.

This invention relates to methods of and apparatus for determining the location and depth of geological formations and particularly to the determining of geological folding in these subsurface formations. In the present method use is made of waves which are transmitted downward into the earth more nearly vertically than heretofore and are directly reflected from the subsurface layer in question to the earth's surface to a point which is usually a distance away from the source that is substantially less than the depth of the subsurface formation.

Claims allowed - 11.

(748) METHOD OF AND APPARATUS FOR DETECTING THE PRESENCE OF PROFITABLE DEPOSITS IN THE EARTH

Günther Laubmeyer, of Kassel-Wilhelmshohe, Germany.

U. S. Patent 1,843,878. Patent issued February 2, 1932.

The method according to this invention consists of collecting underground air by means of a special apparatus and testing it quantitatively for the existence of certain gaseous substances which are in direct relation with the

deposits. The apparatus is in the form of a closure or lid for borings, the lid having besides an outlet pipe at least one bell-shaped cylinder or the like surrounding the outlet pipe; the diameter of this cylinder is larger than that of the borehole. The construction may contain two cylinders arranged concentrically to the other.

Claims allowed - 6.

INDEX[3]

3 The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the Table of Contents, and the third to the page.

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

EMPLOYEE-TIMEKEEPING SYSTEM AND MECHANICAL PAY-ROLL METHODS AT BRITANNIA MINING AND SMELTING CO. (LTD.), BRITANNIA BEACH, B. C.

BY

Albert E. Keller and E. C. Gillingham

I.C.6622,
May, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

EMPLOYEE-TIMEKEEPING SYSTEM AND MECHANICAL PAY-ROLL METHODS AT BRITANNIA MINING AND SMELTING CO. (LTD.), BRITANNIA BEACH, B. C.[1]

By Albert E. Keller[2] and E. C. Gillingham[3]

This paper describing the timekeeping and pay-roll methods of the Britannia Mining and Smelting Co. (Ltd.) is one of a series being prepared for and published by the United States Bureau of Mines regarding office procedures in connection with mining operations.

The general office of the Britannia Mining and Smelting Co. (Ltd.) is located at Britannia Beach, B. C., a town of about 600 inhabitants at tidewater on the east side of Howe Sound and about 30 miles from Vancouver, B. C. The Beach, as it is locally termed, is a company town consisting largely of about 250 employees of the company and their families. Here is the milling and concentrating plant of over 6,000 tons daily capacity; the main service units comprising the electrical, mechanical, blacksmith and woodworking shops and the local electric-furnace foundry; the central-heating steam plant; the air and electric generating units; the ball-manufacturing plant; the warehouses; and the general store, which serves this community and also acts as distributor for the general stores located elsewhere on the property.

As the Beach is served solely by steamship connection, all supplies are distributed from this point by the company's own railroad lines, which are wholly electrified and which connect all of its outlying mining units. Mine ores are handled by means of a coordinated system of raises and a cable tramway from the open-pit workings, at an elevation of about 4,500 feet, down to the top of the mill, approximately at sea level.

The largest mining community is situated about 3 miles from tidewater at what is known as the Tunnel Camp, on the west side of Britannia Mountain, and at an approximate elevation of 2,100 feet above sea level. This townsite accommodates a population of nearly 1,000 people, and is the center of the company's mining activities. At this point are the engineering offices, boarding houses, central mine warehouse, general store, electrical generating units, service shops, copper-precipitation plant, planing mills, car shops, and central sharpening shops.

At the approximate crest of Britannia Mountain is what is locally known as the Barbara Camp, with about 150 men. This camp is equipped only with a boarding house and small store. Men here are engaged in glory-hole mining and in workings immediately below surface in this area. On the east side of the mountain, and at an elevation of about 3,300 feet is a camp of the same type, known as the Empress Camp, which is equipped to accommodate approximately 60 men. Below the Empress, and on the valley floor at an elevation of about 2,700 feet, lies Victoria Camp, with about 200 men. This camp has a boarding house, consignment-stock warehouse, sawmill, and small general store. The workings here extend from the 1800-foot level to the 3350-foot level, and this is the only section on the property which is essentially a timbered mine, employing square-set and timbered-rill methods.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6622."

2 - Chief mine accountant, U. S. Bureau of Mines.

3 - Chief accountant, Britannia Mining and Smelting Co. (Ltd.), Britannia Beach, B. C.

All of the individual mines are connected underground by a system of main haulage tunnels and shafts.

Mines supervision and engineering control centers in the administrative office at Tunnel Camp, while all office and warehousing control is lodged in the general office at Britannia Beach. Frequent contact--usually daily--by means of personal visits and by telephone make for an easily maintained spirit of cooperation between the operators and the general office.

Until late in 1928 an accounting and warehousing staff of 14 men was maintained in the mine office at the Tunnel Camp to prepare pay rolls, keep the accounts and compile statistical data. Cost folios were published monthly and submitted to the mining operators with such other data as were requested. The trial balance only was forwarded to the general office at Britannia Beach for closing in the general ledger accounts. At that time electrical accounting-machine methods were brought to the attention of the company, and after careful examination it was decided to install them.

From the standpoint of efficiency and cost it seemed logical to consolidate all office forces in the general office at Britannia Beach. The change was accomplished with the interested cooperation of the entire personnel, and this improvement now enables the office to supply all information desired with a despatch not possible under the older method, and with a minimum of expense. Under the present improved arrangement, the mine-office accounting staff located at the Tunnel Camp consists of only five men, including the warehouse personnel.

To illustrate one phase of the present accounting procedure, the timekeeping and pay-roll system has been selected and is shown in detail with cuts of all forms and cards used.

TIMEKEEPING AND PAY-ROLL SYSTEM FOR UNDERGROUND EMPLOYEES

Under the method adopted by the company, the timekeeping and pay-roll system has its inception in the employment office at Britannia Beach.

Persons seeking jobs call at the employment office for an interview with the employment clerk. If men are needed, and the applicant appears desirable, the employment clerk sends him to the company physician for physical examination. The physician fills out forms 1 and 2, Physical Examination Record and Examination Certificate, respectively. Form 1 (fig. 1) is retained by the physician for his files, but form 2 (fig. 2) is handed to the applicant for presentation to the employment clerk. If the physical examination is satisfactory, form 3, Employment Card (fig. 2) is then filled out by the employment clerk at the general office, who sends a copy to the mine office.

The employment clerk furnishes the newly employed man with form 4 (fig. 2) and also a brass identification check bearing his pay-roll number. Form 4, Wage Authority, is presented to the timekeeper at the mine office as authority for placing him on the pay roll.

The brass check itself is not made a part of the daily timekeeping system, but is used merely for identification purposes. Neither is the usual cash deposit required when the brass check, bearing the pay-roll number, is issued to the employee upon entering the service of the company.

All employees enter the mine through the shift bosses' office at the tunnel portal. As they pass the inside wicket in the shift bosses' office they call out their pay-roll number to the clerk, who enters it on the upper half of form 5 (fig. 3), Shift On, which shows the pay-roll number of each man going underground as having checked in. As the men come off shift, they pass the outside wicket and call their numbers to the clerk, who enters them in the lower half of form 5, Shift Off (fig. 3). In this manner all men who entered the mine are accounted for.

The check-on and check-off system, described above, in which the clerk participates, applies only to the day shift and the beginning of the afternoon shift at 3 o'clock. The

BRITANNIA MINING AND SMELTING CO., LIMITED

PHYSICAL EXAMINATION RECORD

DATE

Name	Age	Sex	Weight	Wt. Yr. Ago	Height

Nature of Work

Born Year Date	Nationality	Married Single	Widower	Mother D. of Father D. of

Scarlet Diphtheria	Typhoid	Injuries	Indigestion	Rheumatism Neuritis
Tonsillitis Sore Throat	Kidneys Diabetes		Constipation	Menstrual
Influenza Colds	Heart	Operations Abd.	Nervous Breakdown	Alcohol Tobacco
T. B Fam. Hist.	Cough Hemoptysis	Tonsils		Lost time from sickness during last two years
Smallpox Vaccination	Measles Whooping Cough	Rupture	Fainting Spells	

Other points in history

Vision Without Glasses R 20/ L 20/ | With Glasses R 20/ L 20/ | Sym. of Eye Strain | Prev. Exams.

Hearing R L | Ext. Ear | Neck (Thyroid)

Temp. | Tongue | Gums | Treatment for Goitre

Nose Obstruction | Throat (Tonsils)

Teeth R	8	7	6	5	4	3	2	1	1	2	3	4	5	6	7	8	L	Perfect Good Repair
	8	7	6	5	4	3	2	1	1	2	3	4	5	6	7	8		

Caries slight Caries marked | Denture Vacancies | Pyorrhoea Ext. Indicated | Need Cleaning

Heart: Rhythm Murmurs Quality of Sounds | Pulse: Quiet Exercise | B. P.: S. D.

(Cyanosis, Dyspnea, Edema)

* Note—Mark tooth O, if Capped or Pivot; I, if missing; X, if Carious; =, if false.

Form 1 (front)

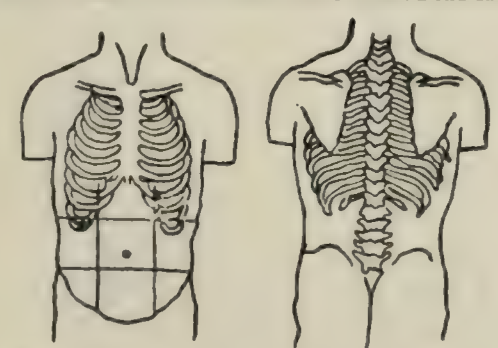
Arteries | Varicose Veins | Hemorrhoids

Hernia | Truss | Footstrain | Spine (Mobility) | Joints

Lungs

Abdomen

Urinalysis Sp. Gr. | Albumen | Sugar | Microscopic and Remarks



"Britannia Mining and Smelting Co. Limited

I hand you herewith a medical report on my physical condition to be used for your information and consideration in connection with my application for employment.

Please retain this report on your files.

DATE

.... Applicant."

77631 G.A.R.LTD.

G. U. and Venereal (Wasserman)

Pupils N. S. Romberg | K. J. Tremour

Nourishment Posture | Gait Skin

(Mark correctable conditions with C.)

IMPAIRMENTS

REMARKS

RATING Physical

Mental

Examined by M.D.

Form 1 (back)

Figure 1.- Physical examination record

Britannia Mining and Smelting Co., Limited

Date ______________ 19 ___

Employment Department,
Britannia Beach, B. C.

As requested, I have examined applicant ______________

______________, and in my opinion he is physically able to perform the duties mentioned

REMARKS

Form 2

Form 3 (front)

Date Hired	Occupation	Date Leaving	Remarks

Form 3 (back)

Britannia Beach, B. C. ______________ 19___

To the Timekeeper

______________ Camp

The bearer ______________

has been employed as ______________ Rate ______

Remarks ______________

BRITANNIA MINING AND SMELTING CO., LIMITED

PER ______________

Form 4

Figure 2.- Doctor's report, employment card, and wage authority forms

Britannia Mining and Smelting Co., Limited

MINE... .... SHIFT ON Time........................ Date.............................19.......

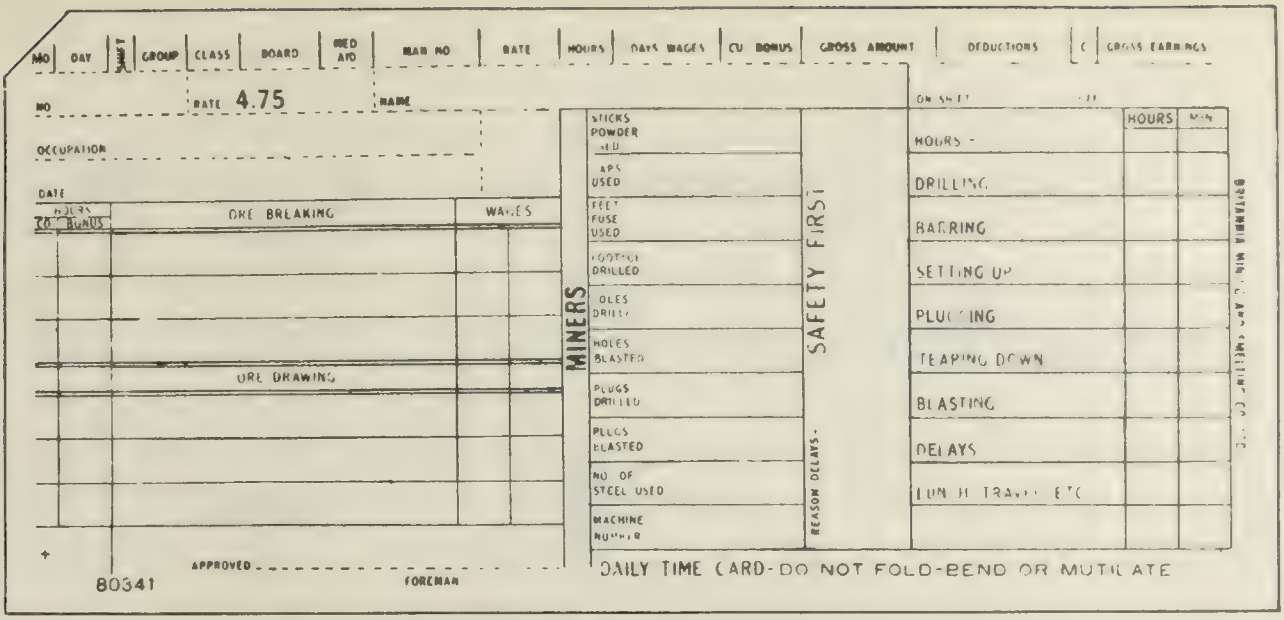
100 to 500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000	2100	2200	2300	2400	2500	2600	2700	2800	2900	3000	

SHIFT OFF Time........................ Date ....19.......

100 to 500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800	1900	2000	2100	2200	2300	2400	2500	2600	2700	2800	2900	3000	

CHECKED ON BY .... CHECKED OFF BY ....

Figure 3.- Form 5 for recording number of men going on and off shift



NO. | DAY | SHIFT | GROUP | CLASS | BOARD | MED AID | MAN NO | RATE | HOURS | DAYS WAGES | CU BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS

NO. RATE 4.75 NAME ON SHIFT OFF

OCCUPATION

DATE

HOURS CO. BONUS | ORE BREAKING | WAGES

ORE DRAWING

APPROVED FOREMAN

80341

MINERS

STICKS POWDER USED
CAPS USED
FEET FUSE USED
FOOTAGE DRILLED
HOLES DRILLED
HOLES BLASTED
PLUGS DRILLED
PLUGS BLASTED
NO. OF STEEL USED
MACHINE NUMBER

SAFETY FIRST

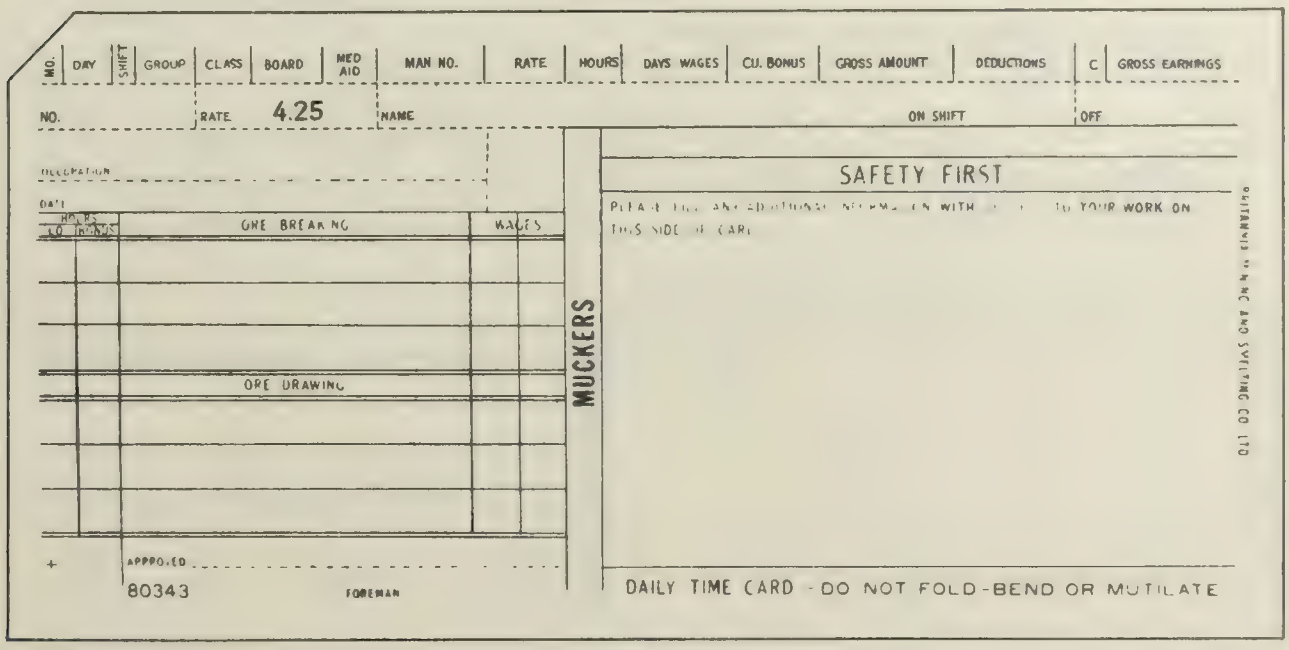
REASON DELAYS

	HOURS	MIN
HOURS -		
DRILLING		
BARRING		
SETTING UP		
PLUGGING		
TEARING DOWN		
BLASTING		
DELAYS		
LUNCH TRAVEL ETC		

BRITANNIA MINING AND SMELTING CO LTD

DAILY TIME CARD - DO NOT FOLD-BEND OR MUTILATE

Form 6



NO. | DAY | SHIFT | GROUP | CLASS | BOARD | MED AID | MAN NO. | RATE | HOURS | DAYS WAGES | CU. BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS

NO. RATE 4.25 NAME ON SHIFT OFF

OCCUPATION

DATE

HOURS CO. BONUS | ORE BREAKING | WAGES

ORE DRAWING

APPROVED FOREMAN

80343

MUCKERS

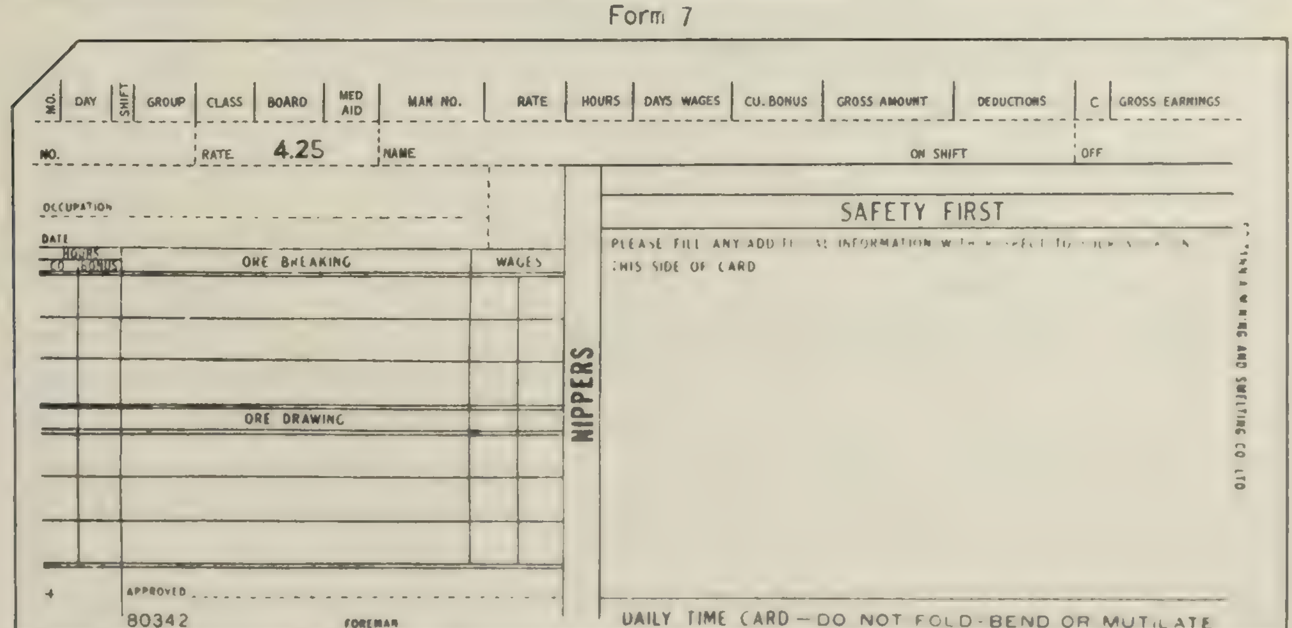
SAFETY FIRST

PLEASE FILL ANY ADDITIONAL INFORMATION WITH RESPECT TO YOUR WORK ON THIS SIDE OF CARD

BRITANNIA MINING AND SMELTING CO LTD

DAILY TIME CARD - DO NOT FOLD-BEND OR MUTILATE

Form 7



NO. | DAY | SHIFT | GROUP | CLASS | BOARD | MED AID | MAN NO. | RATE | HOURS | DAYS WAGES | CU. BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS

NO. RATE 4.25 NAME ON SHIFT OFF

OCCUPATION

DATE

HOURS CO. BONUS | ORE BREAKING | WAGES

ORE DRAWING

APPROVED FOREMAN

80342

NIPPERS

SAFETY FIRST

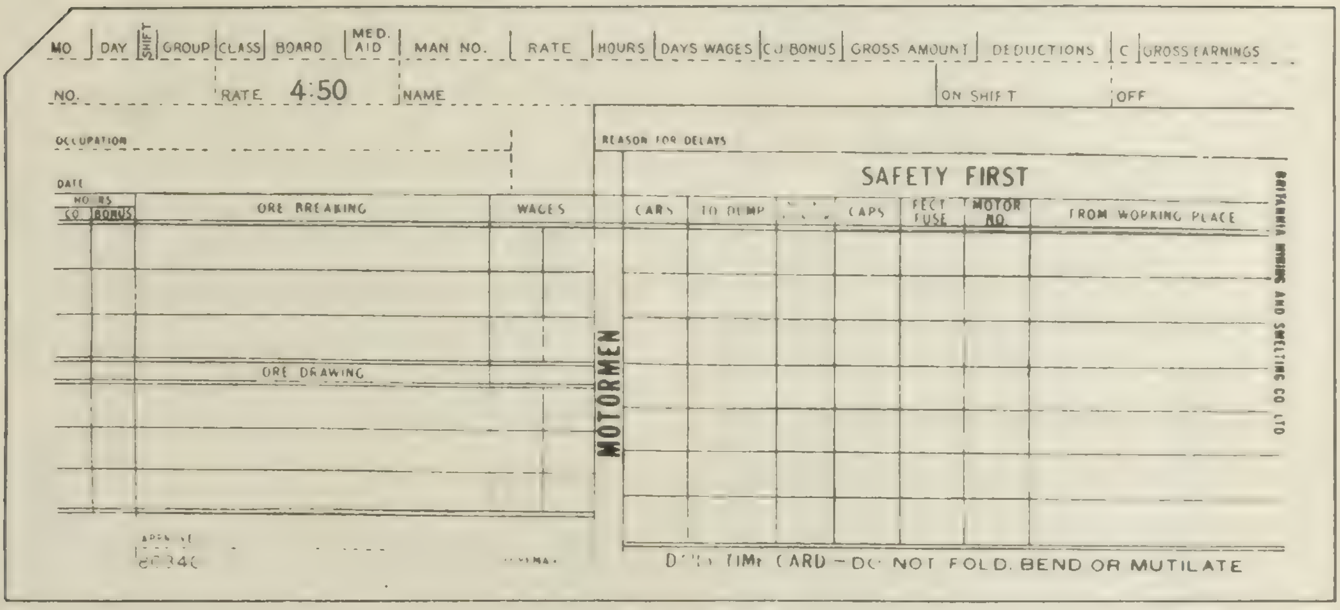
PLEASE FILL ANY ADDITIONAL INFORMATION WITH RESPECT TO YOUR WORK ON THIS SIDE OF CARD

BRITANNIA MINING AND SMELTING CO LTD

DAILY TIME CARD — DO NOT FOLD-BEND OR MUTILATE

Form 8

Figure 4.- Miner's, mucker's, and nipper's individual time cards


MO | DAY | SHIFT | GROUP | CLASS | BOARD | MED. AID | MAN NO. | RATE | HOURS | DAYS WAGES | CJ BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS

NO. RATE 4·50 NAME ON SHIFT OFF

OCCUPATION

REASON FOR DELAYS

DATE

SAFETY FIRST

HOURS CO	BONUS	ORE BREAKING	WAGES

ORE DRAWING

MOTORMEN

CARS	TO DUMP	[illegible]	CAPS	FECT FUSE	MOTOR NO.	FROM WORKING PLACE

BRITANNIA MINING AND SMELTING CO LTD

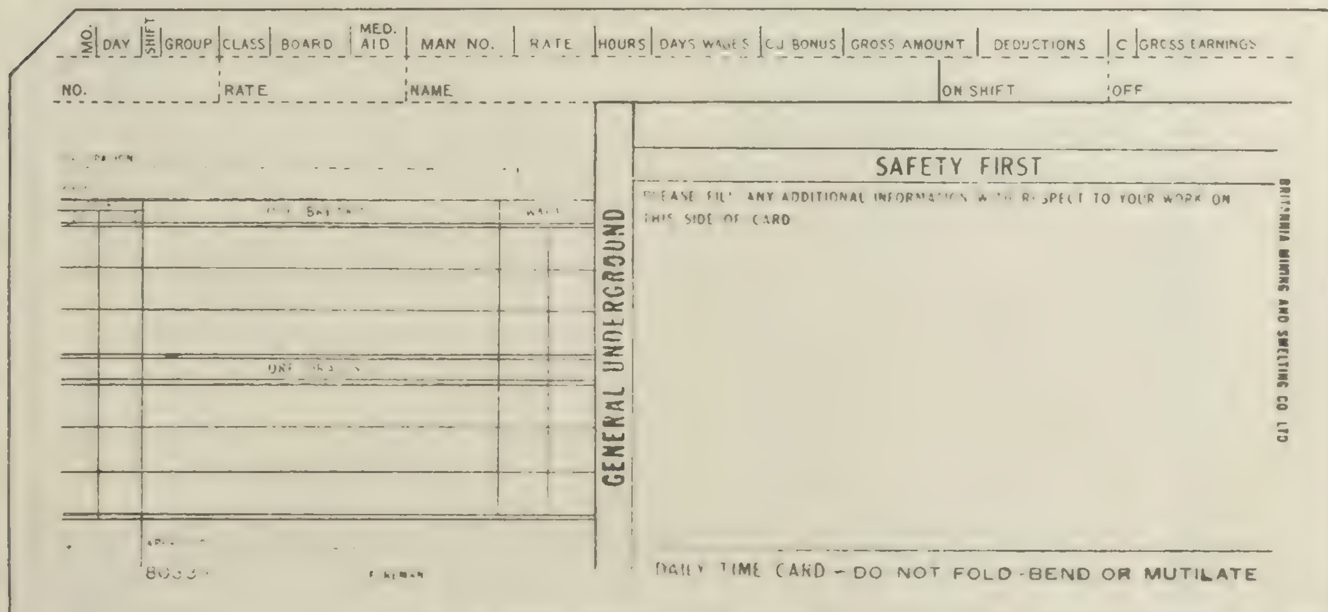
APPROVED

80340

FOREMAN

DAILY TIME CARD – DO NOT FOLD, BEND OR MUTILATE

Form 9


MO. | DAY | SHIFT | GROUP | CLASS | BOARD | MED. AID | MAN NO. | RATE | HOURS | DAYS WAGES | CJ BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS

NO. RATE NAME ON SHIFT OFF

OCCUPATION

ORE BREAKING WAGES

ORE DRAWING

GENERAL UNDERGROUND

SAFETY FIRST

PLEASE FILL ANY ADDITIONAL INFORMATION WITH RESPECT TO YOUR WORK ON THIS SIDE OF CARD

BRITANNIA MINING AND SMELTING CO LTD

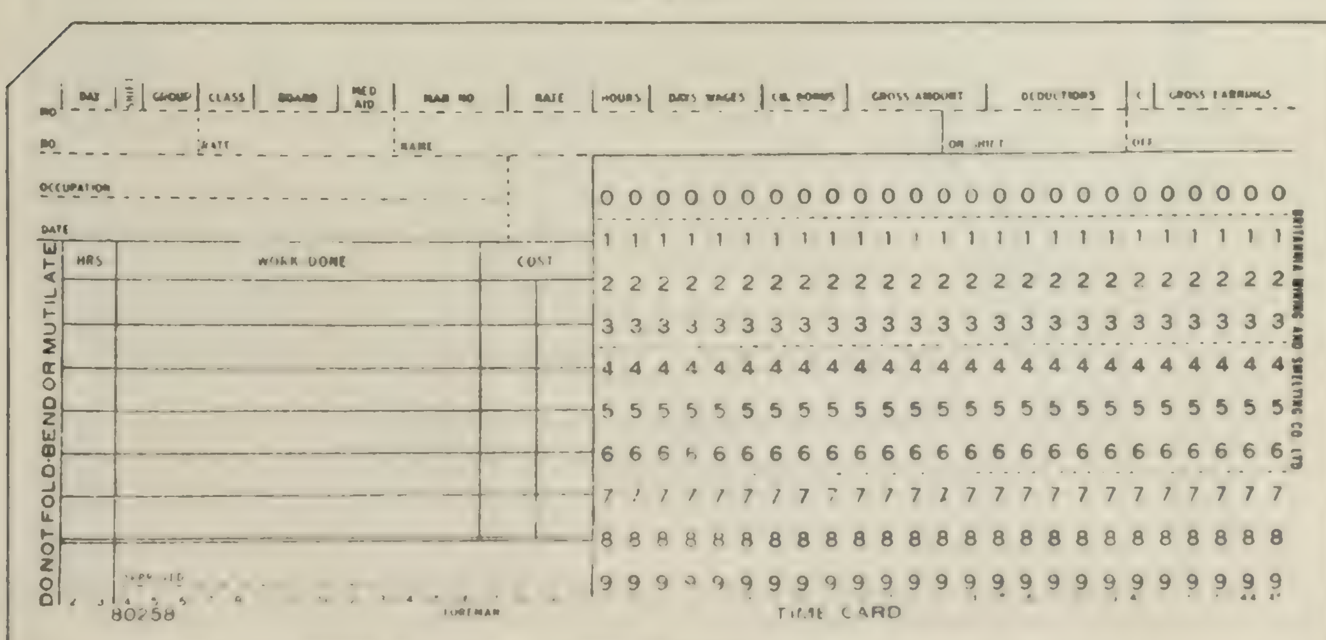
APPROVED

8033

FOREMAN

DAILY TIME CARD – DO NOT FOLD-BEND OR MUTILATE

Form 10


MO | DAY | SHIFT | GROUP | CLASS | BOARD | MED AID | MAN NO | RATE | HOURS | DAYS WAGES | CJL BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS

NO RATE NAME ON SHIFT OFF

OCCUPATION

DATE

HRS	WORK DONE	COST

DO NOT FOLD-BEND OR MUTILATE

0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0
1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1
2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2
3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3
4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4
5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5
6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6
7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7
8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8
9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9

BRITANNIA MINING AND SMELTING CO LTD

APPROVED

80258

FOREMAN

TIME CARD

Form 11

Figure 5.- Motorman's, general underground, and overtime individual time cards

BRITANNIA MINING AND SMELTING CO., LIMITED

______ MINE DAILY SHIFT REPORT

Morning / Afternoon / Graveyard SHIFT ______________ 19__

ORE BREAKING: POWDER No. of Sticks	CAPS No. 6	FUSE No.	FUSE Length	FOOTAGE	HOLES Drilled	HOLES Blasted	BOOT LEGS Blasted	PLUGS Drilled	PLUGS Blasted	MINERS Drilling 4.75	MINERS M & B 4.75	LABOR Mucker 4.25	LABOR Nipper 4.25	LABOR Timber Man 5.25	LABOR Timber Helpers 4.75	LABOR Timber Helpers	WORKING
																	TOTALS

FOR OFFICE USE ONLY	ORE DRAWING: LABOR Miners 4.75	LABOR Muckers 4.25	LABOR Trammers 4.35	LABOR Motor men 4.50	LABOR Brake men 4.25	CARS DRAWN S	CARS DRAWN M	CARS DRAWN L	CARS DRAWN To Transfer Raise	POWDER Kind	POWDER No. of Sticks	CAPS No. 6	FUSE No.	FUSE Length	REMARKS

GENERAL UNDERGROUND

Shift Boss	[illegible] Boss	Machinemen 1st	Machinemen 2nd	Machinemen 3rd	Machinemen 4th	[illegible]	[illegible] Helper	[illegible]	Blaster	Blaster Help	Trackman 4.75	Track Help 4.50	Track Help 4.25	Pipe man	Pipe Help	Crusher Man	Crusher Help	Car Oiler	Safety First	Safety First	Drainage 4.75	Drainage 4.50	Drainage 4.25	Maintenance 4.75	Maintenance 4.50	Maintenance 4.25	[illegible]	[illegible]	Nippers	[illegible]

Shift Boss

Form 12

Figure 6.- Daily shift report form

BRITANNIA MINING AND SMELTING CO., LIMITED
SHIFT RECONCILIATION

	NAME	6 25	5 75	5 25	4 75	4 50	4 25				TOTAL SHIFTS	
	FAIRVIEW BLUFF											
	BARBARA											
	EMPRESS											

Form 13

```
0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0 0
1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1
2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2 2
3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3 3
4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4 4
5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5
    6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6 6
    7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7 7
    8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8 8
    9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9 9
    3 4 5 6 7 8 9 10 11 12 13 14 15 16 17 18 19 20 21 22 23 24 25 26 27 28 29 30 31 32 33 34 35 36 37 38 39 40 41 42 43 44 45
```

Form 14

PAY ROLL NO	RATE AND DEDUCTIONS	HOURS	AMOUNT
379	425	88	04675 *
379	000		

DEDUCTION STATEMENT

M'AID	11	TELE-PHONE	
STORE		ACCTS. REC.	
BOARD	15 75	TELE-GRAMS	
RENT LIGHT	3 00	INCOME TAX	
CLUB	1 00		
COAL WOOD		Total	20 33

CLASS | NUMBER | AMOUNT | CU BONUS | GROSS WAGES

CHECK NUMBER 19707

BRITANNIA BEACH, B. C.

PERIOD ENDING SEP 30 1931

PAY ROLL CHECK

VOID ONE YEAR FROM DATE

DO NOT FOLD, BEND OR MUTILATE

ENDORSE ON BACK

PAY TO THE ORDER OF JOHN DOE

TOTAL DEDUCTIONS | CU. BONUS | GROSS WAGES

XX TWENTY SIX & 42/100 Dollars

PAY $ 02642

4675

BRITANNIA MINING AND SMELTING CO., LIMITED

CANADA EXCISE ACCISE 2 CENTS

CANCELLED

CANCELLED

BANKS: DO NOT CANCEL ON RIGHT HALF OF CHECK

NET BALANCE

To the ROYAL BANK OF CANADA
VANCOUVER, B.C.

Form 15

Figure 7.- Shift reconciliation form, pay-roll deduction and labor distribution card, and pay-roll check

shift bosses are responsible for this procedure on the "graveyard" shift and the end of the afternoon shift, which comes off at 11 p.m. It should be mentioned here that all mines operating in British Columbia are required by law to use a check-on and check-off system.

As each employee passes the wicket when going on shift, he picks up from the counter the individual time card representing his own occupation. There are four types of individual time cards representing occupations, and one for general underground work, also a special one for overtime work. The overtime card is used for all occupations, both underground and surface work. For convenience the overtime card is red but the remaining five time cards are white, although they are distinguished by a colored border at the top, as follows:

Miners	Yellow border	Form 6 (fig. 4)
Muckers	Green border	Form 7 do.
Nippers	Light-buff border	Form 8 do.
Motormen	Purple border	Form 9 (fig. 5)
General underground	Dark-buff border	Form 10 do.
Overtime (for all occupations)	All red	Form 11 do.

Under this system of individual time cards, each employee is his own timekeeper. He keeps a complete record of his daily time and also of certain statistical data as provided for on the several cards.

As the employees come off duty they hand their completed cards to their own shift boss, who approves them by signature.

From the data entered on the completed time cards, each shift boss prepares form 12, Daily Shift Report (fig. 6), which, together with the employees' time cards, is delivered to the mine office. The time cards are now assorted as to shift bosses and then further segregated as to wage rates, which are summarized on form 13, Shift Reconciliation (fig. 7). This summary must agree both in total men and total wages with the data recorded on the Daily Shift Report, form 12.

The daily labor distributions, covering all underground work, are prepared on an electric tabulating machine by the Beach office, in accordance with the code numbers written on the daily shift report by the mine office. The mine office procedure with the individual time cards is now completed, and the cards are sent to the Beach office every morning, grouped according to wage rates.

PROCEDURE AT THE GENERAL OFFICE

Upon receipt of the daily shift report and the individual time cards at the general office, the time cards are punched by an electric key punch to supply the following information: Employees's pay-roll number, rate of pay, hours worked, and amount of wages. The cards are then run through the electric sorting machine, which automatically arranges them in numerical sequence according to the employee's pay-roll number. After this step, the cards are run through the electric tabulator, which totals the amount of wages and shifts, both of which must agree with the Shift Reconciliation, form 13.

The cards are then filed numerically as to pay-roll number, until the end of the semimonthly pay-roll period, at which time they are withdrawn from the files and run through the combination electric listing and tabulating machine, which automatically totals the daily wages of each employee from the individual time cards as received.

Deductions are entered on the face of Pay-roll Check, form 15, in the section designated Deduction Statement. As quickly as this statement is totalled, the Pay-roll Deduction Card,

form 14 (fig. 7), is punched, showing the employee's pay-roll number, the total of deductions and the complement of that figure. The employment of the complement is for automatic machine subtraction of the deductions from the gross wages. The deduction card is then filed behind the last daily time card in the employee's group of cards for the pay period.

The pay roll is now ready to be run. Cards are placed in the hopper head of the electric listing and tabulating machine in pay-roll order. The pay-roll Checks, from 15 (fig. 7), which are printed five on a sheet and separated by pinhole perforations, are fed into the tabulating end of this machine in duplicate. The duplicate copy of the pay-roll checks becomes the Pay-roll Sheet, form 16 (fig. 8). This is a solid sheet without any perforations and is retained in the general office as the permanent pay-roll record.

The operation mechanically is this--the machine is so constructed with controls that when put into operation, it runs through each employee's cards and deduction card and stops automatically at the feed end. The tabulating section of the machine, still in operation, automatically expresses the pay-roll number. total hours, rate of pay, gross wages in total, total deductions, and net wages. This operation expresses one check. The control is touched and the operation begins over again with the next group of cards.

The pay-roll checks and the carbon pay-roll sheet are now put through the special typewriter for typing the employee's name in block type and the amount in pin type. After this has been done the Dominion excise stamps are affixed on all pay-roll checks in excess of $5.

The pay-roll checks are now ready for signature, after which they are separated and grouped according to departments--each department's checks being arranged according to pay-roll numbers in order to facilitate delivery.

Form 17, Change of Rate and Transfer Form (fig. 9), is made out by the shift boss or foreman when the rate of pay of an employee is increased or decreased through occupational change, or when he is transferred to another department. This form must be sent to the pay-roll department as their authority for the changed rate.

Form 18, Employee's Clearance Report (fig. 9), is filled out by the shift boss or the foreman when an employee leaves the service between regular pay days. The employee surrenders this form to the pay-roll department, which issues a time check, form 20 (fig. 10), to him in full settlement of his wages less the various deductions.

Form 20. Time Check (fig. 10), as stated in the previous paragraph, is issued to employees who leave the service between pay days. This form is not negotiable nor does it clear through the bank. For convenience all time checks must be cashed at the company store because there is no regular banking system at Britannia Beach. Twice a month the general office issues a pay-roll check to the company store for the redemption of time checks cashed by them during the previous period. By special arrangement, the Vancouver Bank, with which the company deposits its funds, sends tellers to the property twice a month for a period of about four days immediately after each pay period.

TIMEKEEPING SYSTEM USED IN ALL SURFACE DEPARTMENTS

The surface employees do not check in at the beginning of the shift, nor do they check off at the end.

Two individual time cards, one white and one blue, are used for all surface employees. The blue one, form 21 (fig. 10), applies to hotel employees: and the white one, form 22 (fig. 10), is used for all general surface employees. In addition to these two cards, the timekeeping system for work above ground also includes form 11, Overtime Card, which is the same as that used for underground overtime work.

At the end of the shift the employees deliver their completed time cards to their own foreman, who approves them by signature. On the following morning the foreman delives the

ENDORSE ON BACK

PAY ROLL NO.	RATE AND DEDUCTIONS	HOURS	AMOUNT
379	425	88	04675
379	000		

DEDUCTION STATEMENT

M AID	11	TELE-PHONE	
STORE		ACCTS. REC.	
BOARD	15 75	TELE-GRAMS	
RENT LIGHT	3 00	INC ME TAX	47
CLUB	1 00		
COAL WOOD		Total	20 33

CLASS | NUMBER | AMOUNT | CU. BONUS | GROSS WAGES

CHECK NUMBER 44707

CANCELLED

BRITANNIA BEACH, B. C.

FOR PERIOD ENDING SEP 30 1931

PAY ROLL CHECK

VOID ONE YEAR FROM DATE

DO NOT FOLD, BEND OR MUTILATE

PAY TO THE ORDER OF JOHN DOE

Dollars

PAY $	TOTAL DEDUCTIONS	CU. BONUS	GROSS WAGES
02642	2033		4675

BRITANNIA MINING AND SMELTING CO., LIMITED

CANCELLED

BANKS: DO NOT CANCEL ON RIGHT HALF OF CHECK

NET BALANCE

To the ROYAL BANK OF CANADA
VANCOUVER, B.C.

ENDORSE ON BACK

PAY ROLL NO.	RATE AND DEDUCTIONS	HOURS	AMOUNT
382	375	56	02625
382	450	44	02475
382	000		

DEDUCTION STATEMENT

M AID		TELE-PHONE	
STORE		ACCTS. REC.	
BOARD		TELE-GRAMS	
RENT LIGHT		INC ME TAX	
CLUB			
COAL WOOD		Total	

CLASS | NUMBER | AMOUNT | CU. BONUS | GROSS WAGES

CHECK NUMBER 44708

CANCELLED

BRITANNIA BEACH, B. C.

FOR PERIOD ENDING SEP 30 1931

PAY ROLL CHECK

VOID ONE YEAR FROM DATE

DO NOT FOLD, BEND OR MUTILATE

PAY TO THE ORDER OF

Dollars

PAY $	TOTAL DEDUCTIONS	CU. BONUS	GROSS WAGES
03298	1802		5100

BRITANNIA MINING AND SMELTING CO., LIMITED

CANCELLED

BANKS: DO NOT CANCEL ON RIGHT HALF OF CHECK

NET BALANCE

To the ROYAL BANK OF CANADA
VANCOUVER, B.C.

ENDORSE ON BACK

PAY ROLL NO.	RATE AND DEDUCTIONS	HOURS	AMOUNT
394	330	112	04620
394	370	8	00370
394	000		

DEDUCTION STATEMENT

M AID		TELE-PHONE	
STORE		ACCTS. REC.	
BOARD		TELE-GRAMS	
RENT LIGHT		INC ME TAX	
CLUB			
COAL WOOD		Total	

CLASS | NUMBER | AMOUNT | CU. BONUS | GROSS WAGES

CHECK NUMBER 44706

CANCELLED

BRITANNIA BEACH, B. C.

FOR PERIOD ENDING SEP 30 1931

PAY ROLL CHECK

VOID ONE YEAR FROM DATE

DO NOT FOLD, BEND OR MUTILATE

PAY TO THE ORDER OF

Dollars

PAY $	TOTAL DEDUCTIONS	CU. BONUS	GROSS WAGES
01986	3004		4990

BRITANNIA MINING AND SMELTING CO., LIMITED

CANCELLED

BANKS: DO NOT CANCEL ON RIGHT HALF OF CHECK

NET BALANCE

To the ROYAL BANK OF CANADA
VANCOUVER, B.C.

Form 16

Figure 8.- Pay-roll sheet

This Form to be made out by Shifter or Foreman and turned into office

Britannia Mining and Smelting Co., Ltd.

Change of Rate and Transfer Form

P. R. No.________ Date________________19____

The Bearer ____________________________has been

transferred from ____________________________

to ____________________________

CHANGE IN RATE

His new rate will be $__________ effective__________19____

Transferred by__________________

Transferred to __________________

Form 58 -1M-10-30-2910 G

Form 17

Britannia Mining and Smelting Co., Limited

EMPLOYEE'S CLEARANCE REPORT

Date..............................193......

Name.. No.....................

Occupation ....

Date of last shift worked..............................

Number of Hours worked last shift..........................

Has Wrench been returned?..............................

Signed..
Foreman

Board $.......... ....
Boarding House

Store Account $..............................

Form 18

BRITANNIA MINING AND SMELTING CO., LTD.

SURFACE SHIFT RECONCILIATION

	6.75	6.50	6.25	6.00	5.75	5.50	5.25	5.00	4.75	4.50	4.25	4.00	2.9[illegible]	3.65	3.40	3.15	3.00	2.75	2.50	[illegible] PER MO	[illegible] PER MO	[illegible] PER MO	[illegible] PER MO	[illegible] PER MO					TOTAL SHIFTS	TOTAL MONEY
Steel Shop																														
Machinists																														
2700 Shop																														
Crushers																														
U. G. Mech.																														
Elect. Dept.																														
Power Prod.																														
Ore Transportation																														
3250																														
Carpenters																														
Surface, Whites																														
Surface, Japs																														
Hand Supplies																														
Dia. Drilling																														
Cu. Precip.																														
Sawmill																														
Logging																														
Hotels, Hosp.																														
TOTAL																														

Form 19

Figure 9.- Change of rate and transfer, employee's clearance report, and surface shift reconciliation forms

time cards to the office, where they are posted to form 19, Surface Shift Reconciliation (fig. 9), according to occupation and rates.

The balance of the procedure in connection with the pay-roll system is the same as that used for underground employees.

The labor distribution charges on all surface time cards are designated by operating-account numbers, which are pencilled on the time cards by the employees. These distributions are checked by the foremen as they sign the time cards. The coded account numbers are then punched on Labor Distribution Cards, form 14, by the electric key punch. The labor distribution card has the same form and style as the pay-roll deduction card. This is a stock form of blank statistical card which may be used for many kinds of information. All that is necessary to classify the desired information in the office is to prepare guide cards which allocate certain divisions on the blank card, of which there are 45. Obviously this stock card eliminates the cost of several kinds of special cards. However, the time cards used by the employees must be specially printed.

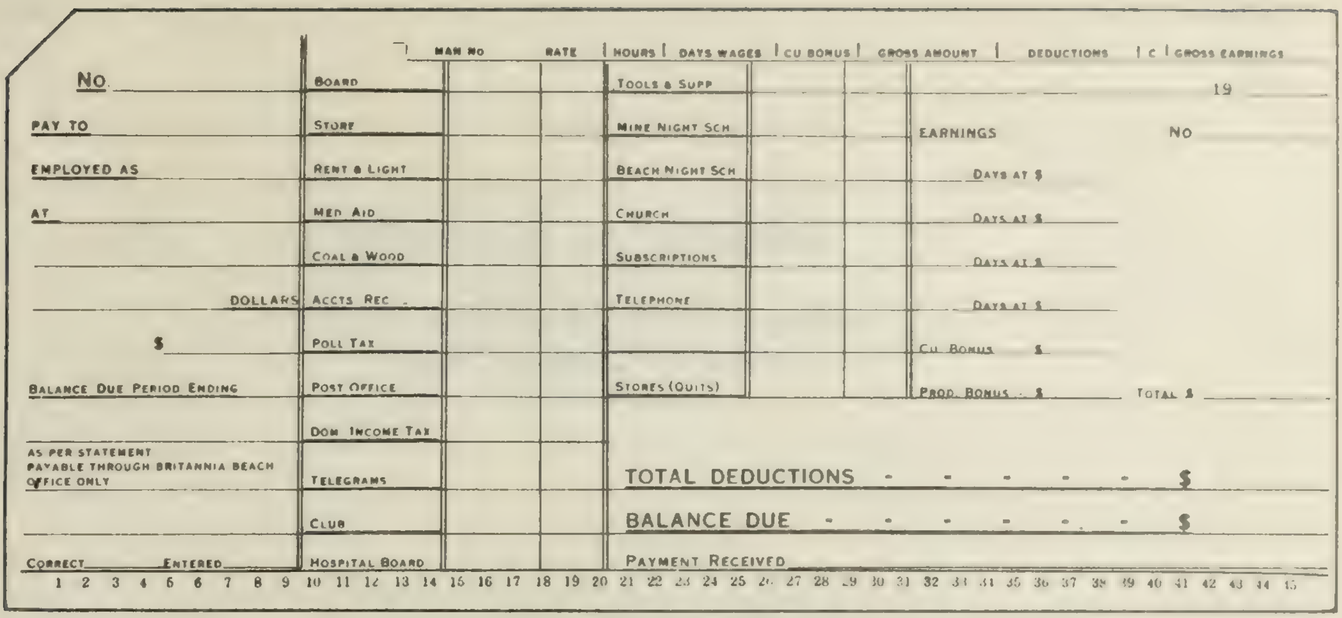
The daily distribution cards which have been punched are then machine-sorted in order of their code numbers and filed with the previous day's cards until they are needed for preparing the cost statements or other accounting and statistical information.

OTHER WORK OF THE ELECTRIC TABULATING MACHINE

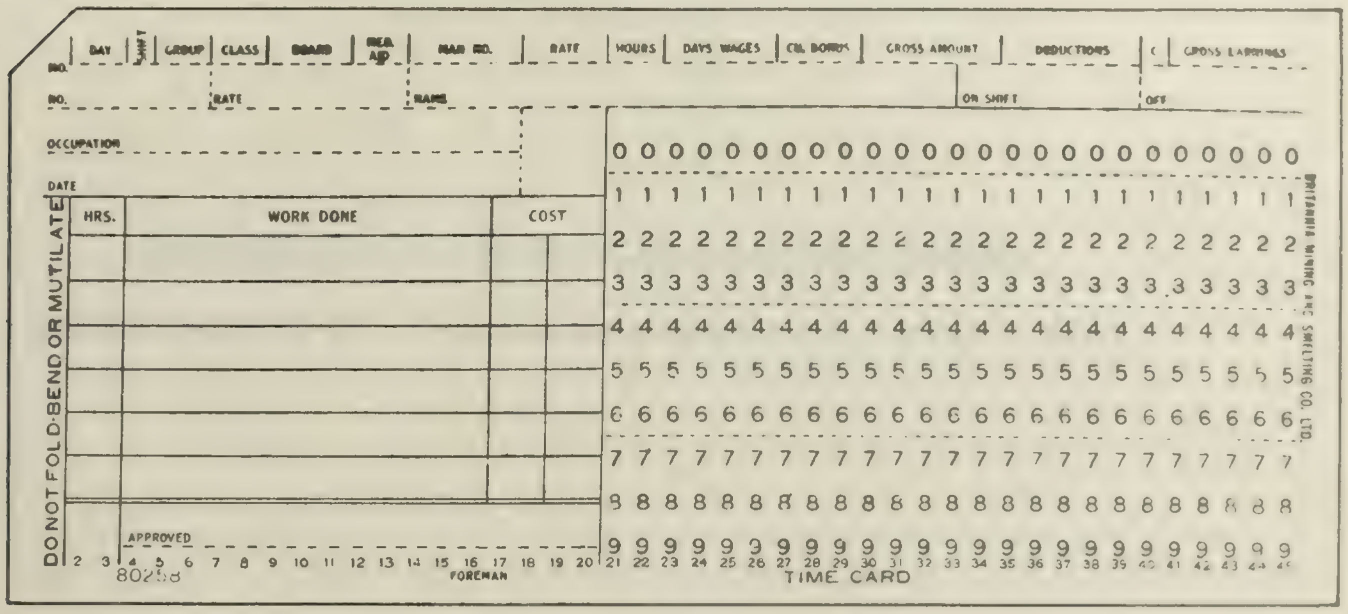
Besides handling records as described, the electric tabulating machine is also used for recording the distribution of warehouse supplies issued during the month. The card used for this purpose is the original requisition card, of dual type, of the same size as the pay-roll card. The left portion for 5 inches is prepared for written information and the remainder of the card is set up for code punching both for account number and amount. At the end of the day both warehouses forward their cards to the general office, where the code numbers and amounts are punched, and where they are assembled and handled similarly to the methods described for the pay-roll system.

The company also employs this tabulating-machine system for other accounting and statistical purposes; in fact, new uses are constantly being found for this machine whereby it quickly assembles various statistics and expedites the office work.

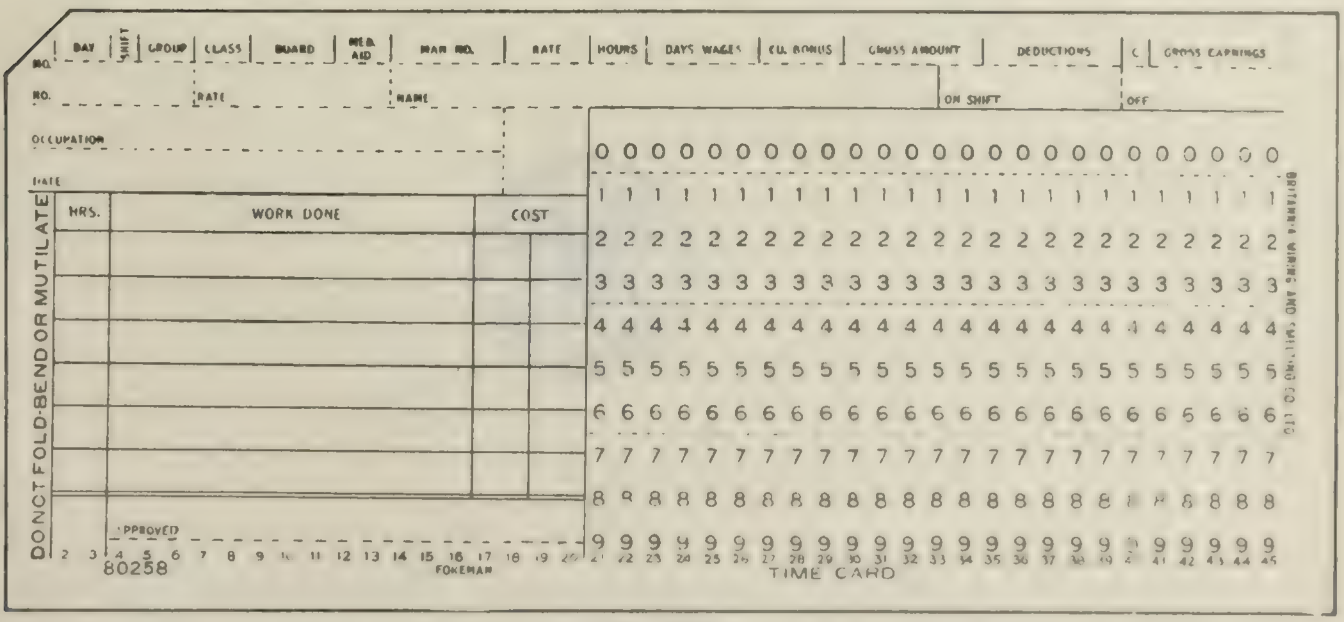
LIST OF TIMEKEEPING AND PAY-ROLL FORMS


No.
PAY TO
EMPLOYED AS
AT
DOLLARS
$
BALANCE DUE PERIOD ENDING
AS PER STATEMENT PAYABLE THROUGH BRITANNIA BEACH OFFICE ONLY
CORRECT ENTERED
MAN NO | RATE | HOURS | DAYS WAGES | CU BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS
BOARD
STORE
RENT & LIGHT
MED AID
COAL & WOOD
ACCTS REC
POLL TAX
POST OFFICE
DOM INCOME TAX
TELEGRAMS
CLUB
HOSPITAL BOARD
TOOLS & SUPP
MINE NIGHT SCH
BEACH NIGHT SCH
CHURCH
SUBSCRIPTIONS
TELEPHONE
STORES (QUITS)
19
EARNINGS
NO
DAYS AT $
DAYS AT $
DAYS AT $
DAYS AT $
CU BONUS $
PROD. BONUS $
TOTAL $
TOTAL DEDUCTIONS - - - - - - $
BALANCE DUE - - - - - - $
PAYMENT RECEIVED

Form 20


NO. | DAY | SHIFT | GROUP | CLASS | BOARD | MED. AID | MAN NO. | RATE | HOURS | DAYS WAGES | CU. BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS
NO. RATE NAME ON SHIFT OFF
OCCUPATION
DATE
HRS. | WORK DONE | COST
APPROVED
FOREMAN
80258
DO NOT FOLD-BEND OR MUTILATE
BRITANNIA MINING AND SMELTING CO. LTD.
TIME CARD

Form 21


NO. | DAY | SHIFT | GROUP | CLASS | BOARD | MED. AID | MAN NO. | RATE | HOURS | DAYS WAGES | CU. BONUS | GROSS AMOUNT | DEDUCTIONS | C | GROSS EARNINGS
NO. RATE NAME ON SHIFT OFF
OCCUPATION
DATE
HRS. | WORK DONE | COST
APPROVED
FOREMAN
80258
DO NOT FOLD-BEND OR MUTILATE
BRITANNIA MINING AND SMELTING CO. LTD.
TIME CARD

Form 22

Figure 10.- Time check (discharge check) card, and hotel and general surface individual time cards

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

PROCEDURE OF THE PURCHASING AND SUPPLY DEPARTMENTS OF THE MIAMI COPPER CO., MIAMI, ARIZ.

BY

FRED L. BISHOP AND ALBERT E. KELLER

I.C.6623,
May, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PROCEDURE OF THE PURCHASING AND SUPPLY DEPARTMENTS OF THE MIAMI COPPER CO., MIAMI, ARIZ.[1]

By Fred L. Bishop[2] and Albert E. Keller[3]

INTRODUCTION

This paper describing the methods employed in the purchasing and supply departments of the Miami Copper Co., Miami, Ariz., is one of a series being prepared for and published by the United States Bureau of Mines on office practice in mining operations.

GENERAL DESCRIPTION

The Miami Copper Co.'s plant is located about 1 mile from the town of Miami, in the Globe-Miami Mining District, Gila County, Ariz.

The property consists of a complete mining plant, concentrator, high-pressure steam power plant, and many other attendant structures. In normal times the company employs about 1,500 men, and mines and mills about 6,000,000 tons of ore annually.

The average inventory investment amounts to approximately $700,000, which turns over about three times a year; that is, in the neighborhood of $2,000,000 worth of supplies are purchased and used annually in the operation of the plant.

The procedures of the purchasing department and the supply department are consolidated in this one paper because both departments are in charge of one man, the purchasing agent. This method of operation facilitates the handling of both departments, because the usual gap so often existing between two individual departments is eliminated together with the accompanying loss of time in handling transactions between them. The plan also eliminates the duplication of records and effects a reduction in the combined operating expenses as against those of the individual departments.

The departments have two warehouses, the main one adjoining the plant buildings where it is readily accessible to all plant departments, except the power-house department. The other warehouse is about ¼ mile distant at the industrial-railway yard which connects with a transcontinental railroad. The purchasing and supply offices are in the main warehouse, where they can closely supervise the receiving and issuing of supplies.

The systems used in these two departments are the result of considerable effort in handling the average inventories necessary for a company of this size at a minimum of inventory investment and operating expense.

The printed forms used in this combined system are shown in the section of this paper entitled "Detailed procedure of purchasing department and supply department."

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6623."

2 - Purchasing agent, Miami Copper Co., Miami, Ariz.

3 - Chief mine accountant, U. S. Bureau of Mines.

SUPPLIES

The supplies issued amount to approximately $165,000 monthly, and are distributed in more than 400 accounts through an average of 3,600 individual supply orders on the storekeeper each month. These orders do not include the daily issuance of such quantity items as fuel oil, flotation reagents, drill steel, and grinding balls.

The flotation reagents, drill steel, and grinding balls are charged out at the end of the month, but the fuel oil used is charged out semi-monthly. This method, of course, eliminates the necessity of handling numerous individual supply orders, which is a big factor in reducing them to only 3,600 monthly.

This volume of business with its attendant procedures is handled by 23 people, five of whom, including the purchasing agent, operate the office. The remaining 18 persons are employed in the warehouse building and in the outside storage yards, the work at which includes trucking from the railroad depot to the warehouse and also the delivery service to the various departments. Two men are helpers on trucks and eight form an unloading and loading crew. These eight men spend considerable time in unloading timber from railway cars and reloading it on mine cars for transporting underground. They also handle other carload items such as steel, pipe, grinding balls, machinery, xanthate, cement, etc. Sometimes, however, it is temporarily necessary to increase this crew when deliveries from the railroad company are above normal. This temporary increase eliminates congestion in the company's yard and the accrual of demurrage charges.

Classifications

The warehouse supplies are grouped into the following 23 classifications with a separate account in the ledger for each:

- Air-drill stock
- Bolts, nuts, washers and rivets
- Brick and cement
- Castings
- Coal and coke
- Drill steel
- Electrical supplies
- Explosives
- Flotation reagents
- Gasoline and oils
- Grinding balls
- Iron and steel
- Lumber
- Machinery
- Machinery parts
- Miscellaneous supplies
- Packing, waste, hose and belting
- Painters' supplies
- Pipe and fittings
- Plumbers' supplies
- Railway supplies
- Stationery and office equipment
- Tools

These classifications are further subdivided into approximately 8,000 items, with a separate loose-leaf ledger sheet, form 10 (fig. 14), for each item.

Locations

The warehouse supplies are stored in 16 different locations, as follows:

No.	Location	No.	Location
1	Main warehouse (inside)	4	Second floor annex
2	Pipe and bar steel racks and heavy warehouse	5	Main warehouse (outside)
3	First floor annex	6	Railway yard and lower warehouse

No.	Location	No.	Location
7	Power house	12	Plate shop
8	Mill yard	13	Churn-drill warehouse
9	Mine yard	14	Oil house
10	Carpenter shop	15	Belt and hose house
11	Saw-mill stock	16	Old oil house

The reason for storing supplies in so many different locations is for convenience in handling from storage to point of consumption. This is explained as follows:

Location No.	Storage Contents
1	Frequent-moving and small items
2	Pipe, steel, and heavy items
3	Principally machinery parts
4	Do.
5	Heavy castings
6	Timber, large pipe, cement, hoist ropes, fuse, carbide, and other bulky items, all of which are handled advantageously from that point
7 to 13, inclusive	Heavy material used exclusively by the respective departments
14	Oils, gasoline and inflammable material
15	Rubber goods
16	Obsolete items

Accounting

When posting from an invoice to form 10, the location of the material is entered in the "location" square in the lower right corner of the form. This reference is made because the supplies are in many different places over a considerable area; by referring to the ledger sheet they are easily located for inventory purposes, and also by employees who are asked to issue supplies in emergency cases when regular employees are not on shift and who might not be familiar with their location. In storing supplies for a plant of this size, it must be remembered that many items are carried which are necessary to the continuity of operation but which may be seldom used and might be lost track of unless their location is recorded.

For emergency purposes a small quantity of supplies is carried in an underground supply station on the 620-foot level. These are charged out of the warehouse stock at time of transferring underground.

The supply department, under this plan of operation, handles the accounting for deferred charges covering major repairs and replacements. For instance, charges for a $3,000 article might be spread over a period of six months. Accordingly the supply classification would be credited with $500 per month, and the account benefited would be charged a like amount. In this particular illustration, as soon as the $3,000 article is transferred from the warehouse. six separate orders on the storekeeper, form 12, would be written up and placed in a suspense file. Each month thereafter, until all have been used, one of them would be removed from the file and put through the current warehouse supply records in the usual manner.

INVOICES

Invoices, when received, are rubber-stamped to show:

a - Date of receipt of goods,
b - Checking of extensions, and
c - Approval of prices.

The next operation is checking the invoice against the office copy of the order, form 3-1 (fig. 6), by pencil-checking the items invoiced, also showing on rubber stamp impression on form 3-1 a symbol designation, and checking in the "invoice received" column. Then the invoice is checked for extensions, prices, f.o.b. point, and filed with a copy of the order, except in case of discount invoices, in which instance one copy is promptly vouchered. The net invoices are paid 30 days after date. The three copies are kept on the order file until the material has been received and records have been completed on the order, at which time the original and one copy are passed to a "tickler" file, and vouchered by the purchasing department when due. Before vouchering, the "f.o.b. point" is noted, and if transportation charges have not been prepaid on an f.o.b.-destination shipment, deduction of the charges is made on the voucher. Upon arrival of the material, the warehouseman checks it and lists all the items on the receiving report, form 6, which together with the transportation bill, is passed to the office. One copy of each of those forms is attached to the office copy of the invoice. All invoices are numbered consecutively and the related papers bear the invoice number. The invoice is then indexed alphabetically under firm name, on a 3 by 5 inch card and filed numerically. This card record shows the date, invoice number, order number, and amount of invoice.

Payment of Invoices

Under this system of office operation, the invoices are vouchered by the purchasing department, rather than by the accounting or treasury department, as is done by most companies. Likewise any correspondence concerning adjustments, errors, freight deductions, etc., are handled by the purchasing department. All invoices are vouchered on form 15 (fig. 19), in quadruplicate. The original, duplicate, and triplicate are delivered to the Miami accounting office; the original is properly signed there, and mailed to the payee; the duplicate is retained by the Miami accounting office for record and file; the triplicate is mailed to the New York office for its files; and the quadruplicate, which is a copy on blank paper, remains in the office of the purchasing department. The quadruplicate is filed alphabetically and serves as a quick reference for all transactions with each firm. A copy of the invoice, or invoices, being paid, is attached to the duplicate and triplicate copy of the voucher, forms 15-A and 15-B, both of which are approved by the purchasing agent. The voucher checks themselves, all of which are drawn on New York banks, are signed by the general manager and the chief clerk.

CASH DISCOUNTS

The cash discounts deducted from vendors' invoices are not considered a reduction in the cost of supplies purchased. These discounts are allowed in consideration of prepayment of the invoices before they are due; hence the discount is treated as an earning.

OPERATING EXPENSE

The monthly operating expenses of the combined purchasing department and the supply department are absorbed by the mining department and the milling department - half to each. This equal distribution is made because, over a period of years, it has been observed that the supplies used by each of these departments is approximately the same amount.

Figure 1.— Monthly inventory of supplies

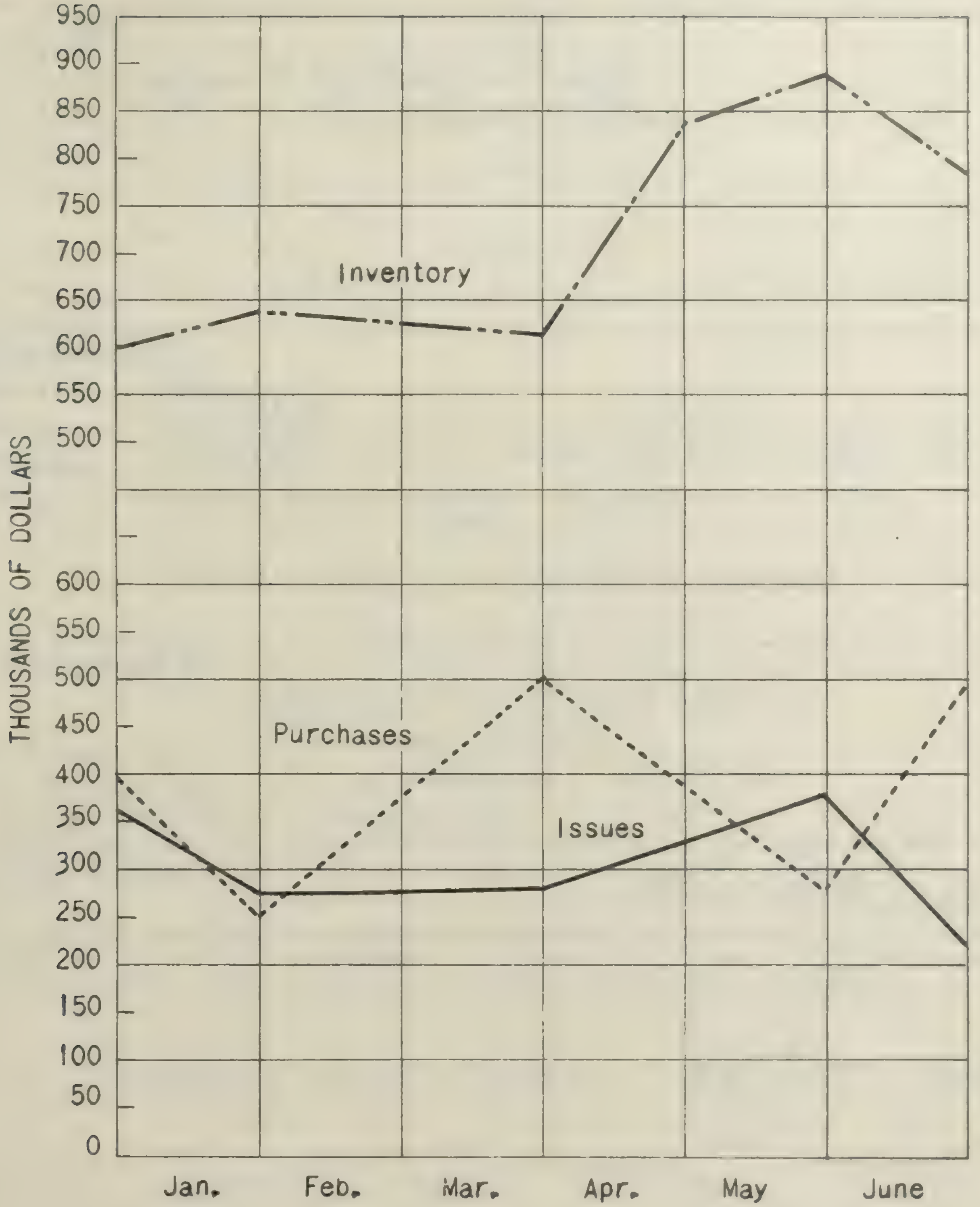


Figure 2.- Monthly supplies purchased and issued

MIAMI COPPER COMPANY

Requisition No. 12642 October 23 1931

Last Order	On Hand	Quantity Wanted	ARTICLES	COST
12089		50	dozen #4 Miners shovels	
			Long handle - Round point	

Stock Item (Yes) (~~No~~) ✓

Ordered by Mine

Required for Gen'l U. G.

Date Required 12-20-31

(Signed) J H Hensley Jr

Form 1

Figure 3.- Requisition on purchasing department

Form 227 2m 6-29 MP&PCo B8796

MIAMI COPPER COMPANY

PURCHASING DEPARTMENT

Miami, Arizona Oct. 23, 1931

Brown Shovel Co.,

Chicago, Ills.

Kindly quote hereon your lowest price on the following material

Each item must be priced separately. Show discounts only when standard lists apply. Your quotation must reach us on or before Nov 1, 1931

J L Bishop
Purchasing Agent

Quantity	MATERIAL	Unit	Price or Discount	Weight
50	Doz #4 Miners' Shovels, L. H., R. P.	Doz	$10.00	3600

Material offered can be shipped from Chicago within 10 days after receipt of order at price quoted F. O. B. Miami Ariz

Invoice subject to discount of 2 per cent for cash in 10 days from date of Invoice

Date 10/27/31 (Sign here) Brown Shovel Co

Form 2

Figure 4.- Inquiry form

Miami Copper Company
Miami, Arizona

_______________ (Date)

We acknowledge receipt of your Order numbered as below

Shipment will be made on _______________ (Date)

Order No. 11772 _______________

Please detach and mail promptly

MIAMI COPPER COMPANY

MIAMI, ARIZONA Oct 30, 1931

Order No. 11772

SHIPPING DIRECTIONS

MIAMI COPPER CO
MIAMI
ARIZONA

Brown Shovel Company,

Chicago, Ill.

Please fill the following order on account of the MIAMI COPPER CO and forward INVOICE IN TRIPLICATE and ORIGINAL bill of lading to this office

If terms are not marked on invoice we will deduct 2% for cash in ten days

30 - Doz. #4 Miners Shovels, L. H., R. P.

Price: $10.00 doz. F.O.B. Miami, Arizona.

Shipment: Ten days from receipt of order.

Terms: 2% 10 days, 30 days net.

SHIP VIA **C.R.I. & P. and Sou. Pac.**

MIAMI COPPER COMPANY,

Purchasing Agent.

12642-JHH

MIAMI COPPER COMPANY

PURCHASING AGENT'S COPY Oct 30, 1931

ORDER NO 11772

Brown Shovel Company,

Chicago, Ill.

30 - Doz. #4 Miners Shovels, L. H., R. P.

Price: $10.00 doz. F.O.B. Miami, Arizona.

Shipment: Ten days from receipt of order.

Terms: 2% 10 days, 30 days net.

C.R.I. & P. and Sou. Pac.

MIAMI COPPER COMPANY,

Purchasing Agent.

40922

12642-JHH

11772

Form 3

Figure 5.- Original and record copy of order form

STATISTICS

For ready reference as to different phases of the business, various graphic charts are prepared which quickly picture useful information over a long period of time. Two of these charts are reproduced. One of them (fig. 2) plots the monthly inventory of supplies, also the monthly supplies purchased and issued. The other one (fig. 1) plots the number of orders, form 12, drawn on the storekeeper, also the number of items covered by the orders. This first chart covers a period of three years.

DETAILED PROCEDURE OF PURCHASING AND SUPPLY DEPARTMENTS

For the purpose of more clearly illustrating the detailed procedure of the combined departments, the following pages outline the procedure in the purchase of 50 dozen No. 40 miners' shovels at $10 per dozen, and in the issue of one dozen of these shovels. All of the forms used in these departments from the time the requisition is made until the shovels are received and some of them issued, including showing the balance on inventory, and complete entries on the forms, are described in this example.

Requisition on Purchasing Department

Form 1 (fig. 3).-It will be noted that form 1, Requisition on Purchasing Department, has been issued for 50 dozen miners' shovels. In the lower left corner it is indicated that these are wanted for stock purposes. By this method the supply department knows that upon arrival these shovels are to be placed in the warehouse stock. The date required is also an important factor, because in many instances requisitions are written for urgent items; but in the case illustrated about two months delivery is acceptable. This form is issued by department heads for their own requirements, which are not controlled by the supply department. On regular stock items the supply department keeps a minimum on hand, this quantity depending on consumption. Form 1 is made in duplicate, the requisitioner retaining the duplicate copy.

Inquiry Form

Form 2 (fig. 4).- As is indicated, this form is sent (in this particular illustration) to the various suppliers of shovels to secure prices, delivery, etc.

Order Form

Form 3 (fig. 5).- When quotations have been received on form 2 and it has been decided from whom the most advantageous purchase can be made, order form 3 is written for the shovels in question. One original and four carbon copies are made of the order. The original is mailed to the vendor; the duplicate copy is retained in the purchasing department office; the triplicate copy is used by the warehouse foreman for the purpose of information as to kind and size of material expected, and also for checking receipt of the material; the quadruplicate copy is sent to the department which requisitioned the material; and the fifth copy is sent to the New York office of the company for their records. On the purchasing department's copy is shown a complete record of receipt of invoice, bill of lading, date of shipment, date of receipt of material, etc., also the invoice number covering the particular shipment. All unfilled orders are kept on a current file until the material is received and all papers have been passed, at which time they are filed numerically in a permanent post binder.

Immediately after the orders are written they are indexed on 3 by 5 inch card records, first under the name of the vendor and then under the name of the item.

B Order Form

Form 3A (fig. 6).- Temporarily deviating from the shovel transaction at this point, attention is called to form 3-A, known as a "B" order. This form is used only for small items purchased from merchants within the Miami district. It is made in triplicate, the original being sent to the vendor, the duplicate copy retained in the purchasing department office for purposes of record, the same as order blank form 3, and the triplicate copy sent to the warehouse foreman for the purpose of checking receipt of material.

Requisition on New York Purchasing Department

Form 3-B (fig. 7).- This form is sent by the Miami office to the New York purchasing department, when requisitioning material for that office to purchase:

(a) Under certain contracts,
(b) Unusually large quantities, or
(c) Material which is produced in the East and can be purchased more advantageously through the New York purchasing department.

This form is made in duplicate, the original is mailed to the New York office, and the duplicate is retained in the Miami purchasing department office.

The New York office furnishes to the Miami office three copies of all orders placed by them. These copies are used for checking against the requisition drawn on the New York office, and for completing the files in the same manner as is done on order form 3.

Acknowledgment Tracer

Form 4 (fig. 8).- When the stub attached to the top of order, form 3, is not promptly mailed by the vendor to acknowledge receipt of the order, a postal card, form 4, is immediately sent to him. This standardized postal card shows whether or not the order was shipped, and at the same time impresses upon the firm the importance of using the acknowledgment stub. This form of tracer for acknowledgment of the shovels ordered is shown in the exhibit.

Tracer for Shipment, or Invoice

Form 5 (fig. 9).- When it is apparent that the vendors are slow in making shipment, or in rendering an invoice, form 5 is mailed to them.

Receiving Report

Form 6 (fig. 10).- Form 6 is made up by the warehouse foreman upon receipt of material. In the exhibit it is made up to cover the 50 dozen shovels ordered on form 3. This report is prepared in duplicate, and the lower part, or inspection report, is executed by the warehouse foreman covering all material that he is qualified to inspect. If, however, the material consists of intricate pieces of machinery, etc., the warehouse foreman ignores the inspection report blank, and both copies of the receiving report are passed into the purchasing department office for noting receipt of material on order form 3.

MIAMI COPPER COMPANY

Order No.
B 11123

Miami, Ariz.

Please furnish the following for our account, delivering same by

Mark on each package the NUMBER OF THIS ORDER and furnish INVOICE IN TRIPLICATE

MIAMI COPPER COMPANY

Purchasing Agent

MIAMI COPPER COMPANY

Order No.
B 11123

Miami, Ariz.

Please furnish the following for our account, delivering same by

Mark on each package the NUMBER OF THIS ORDER and furnish INVOICE IN TRIPLICATE.

[illegible]	INV. REC'D	B-L REC'D	GOODS SHIP'D	GOODS REC'D	VIA	REG. No.	BY	NOTI FIED	REC'D	INVOICE NUMBER

MIAMI COPPER COMPANY

Purchasing Agent

Order No.
B 11123

Form 3A

Figure 6.- Original and second and third copies of B order form

ORIGINAL

MIAMI COPPER COMPANY

MIAMI, ARIZ.,______________________

REQUISITION NO. 9601

PLEASE FURNISH THE FOLLOWING MATERIAL, SHIPPING SAME VIA

DUPLICATE

MIAMI COPPER COMPANY

MIAMI, ARIZ.,______________________

REQUISITION NO. 9601

PLEASE FURNISH THE FOLLOWING MATERIAL, SHIPPING SAME VIA

Form 3B

Figure 7.- Original and duplicate of form for making requisitions on the New York purchasing department

[illegible]

[illegible]

Miami, Arizona.
Date 11/13/31

Gentlemen:

Please refer to our order No. 11772 The stub attached to top of order is intended for you to fill out and return promptly and serves to acknowledge receipt of order, also inform us as to date of shipment.

Please comply with our request on this and future orders

MIAMI COPPER COMPANY

Form 4

Figure 8.- Acknowledgment tracer

Miami, Arizona,

Nov 22, 1931

Gentlemen:

Kindly advise by return mail when we may expect shipment of ~~invoice covering~~ material on our order:
No 11772
50 doz. Shovels

MIAMI COPPER COMPANY,
Purchasing Dept

Form 5

Figure 9.- Tracer for shipment, or invoice

MIAMI COPPER COMPANY

Receiving Report

From Brown Shovel Co. Address Chicago, Ill.

Quantity	Name of Article	Weight
50	Doz. L. H. R. P. Shovels	3600#
	(Stock) Location #2	

Express	Freight	Parcel Post	Order No	Received	Date
	✓		11772	A. G. Philp	12-15-31

INSPECTION REPORT   Date ✓

Above Material, your Reqn. has been placed Furnish "Order on Storekeeper" to cover

Material listed has been examined and found to be

O.K.

NOTE: (If as ordered indicate by your O. K.)
(If not as ordered state particulars)

Examined by: A.G.P.

Form 6

Figure 10.- Receiving report, made in duplicate

Miami, Arizona.

Dec 6-1931

Gentlemen:

Our order plainly specifies that invoices should be rendered in triplicate, accompanied by shipping documents. Your invoice is returned for the following documents indicated by check mark.

✓ BILL OF LADING

_____ EXPRESS RECEIPT

_____ INVOICE IN TRIPLICATE

Your immediate attention will enable us to pay your invoice when due.

MIAMI COPPER COMPANY

Form 7

Figure 11.- Tracer for additional copies of invoice or shipping papers

FUEL OIL UNLOADING REPORT

DATE

INCHES OIL PUMPED TO HILL TANKS

Car Initial	Car Number	Marked Capacity	Corrected Capacity	Sp Gr.	Tem F.	Deg Be'	Deg. Be' c60 F	Space Above or Below Shell	Overage or Shortage Gallons	Remarks

Form 8

Figure 12.- Oil receiving report

MIAMI COPPER COMPANY

Miami, Arizona, 10-24-31

To Purchasing Agent:

I certify that the following tools wanted by Mr. John Brown #1071 will be required for use in his work with the Miami Copper Co.

1-#17 Combination Square with Center Head-18" Blade. L. S Starrett Catalog #24 page 71

1-#4 Screw Pitch Gage. L S Starrett Catalog #24 page 201

Approved: HLB

F. Martin
Department Head

Form 9

Figure 13.- Employee's requisition for tools

MIAMI COPPER CO.
MIAMI ARIZONA

Minimum 150

DEBITS					CREDITS															
INVOICE	REC'D	ORDER	QUANTITY	AMOUNT	DATE	ACC'T	QUANTITY	COST PER Ea	DATE	ACC'T	QUANTITY	COST PER	DATE	ACC'T	QUANTITY	COST PER	DATE	ACC'T	QUANTITY	COST PER
40922	12/15/31	11772	600	500 00			600	.8333												
					12/23	19	12													
					DEC 28 '31 INVENTORY		588													
					JAN 1 '32		588													

LOCATION		CLASS
2		Tools

ARTICLE	DESCRIPTION	
Shovels	#1 Miners' L. H., R. P.	For signal tab

Form 10

Figure 14.- Stock ledger sheet

In the instance of the 50 dozen shovels which have been inspected and approved by the warehouse foreman, the duplicate copy of the receiving report is sent to the original requisitioner, to notify him of arrival of the goods.

In the event that the warehouse foreman does not feel qualified to approve the material received, the original copy of the inspection report is sent to the department head who requisitioned the material, asking him to have it inspected by engineers and to execute and return the report. It is important to follow this method of handling the inspection report, as it guards against accepting material that does not come up to the specifications. The invoice, or shipping manifest, is not used for checking the material into the warehouse.

Tracer for Additional Copies of Invoice or Shipping Papers

Form 7 (fig. 11).- This form is self-explanatory. It is used when sellers do not comply with instructions contained on order blank, form 3. A tracer for bill of lading covering the shovels ordered is also shown.

Oil Receiving Report

Form 8 (fig. 12).- This form is used by the man in charge of receiving and unloading fuel oil, who is directly under the supervision of the purchasing department. All information on this form is filled in by the unloader, except columns marked "Degrees Baumé at 60° F," and "Overage or Shortage - Gallons." Entries in these two columns are made by the clerks in the purchasing department, and are used as a basis of adjustment in case of shortage or overage.

Employees' Requisition for Tools

Form 9 (fig. 13).- It is the policy of the company to issue from warehouse stock, or to buy such tools as employees require in their work. Form 9, after being signed by the department head, is submitted by the employee wanting tools.

Stock Ledger Sheet

Form 10 (fig. 14).- This form is used as a stock record of all material carried in the supply department, with the exception of fuel oil and timber, which, for convenience, are carried on form 11. Form 10 is carried in loose-leaf binders with a separate sheet for each article. The name of the article is written at the bottom of the sheet and is always visible when the book is open.

There is one division of columns for debits, and the remainder of the sheet, printed on both sides, is for credits. This style is necessary because there are more credit (issue) entries than there are debit (purchase) entries. In the exhibit there are shown the transactions involved in handling shovels purchased on form 3, from the entry of receipt of shovels, charging out 12 of them to account 19, to the inventory balance as of January 1, 1932. As a matter of explanation, in the event an additional lot of shovels is purchased at a different price, the average cost price for the entire quantity on hand at any one time would be used in charging out the shovels as issued. This method is more satisfactory than the "oldest-cost-first" plan.

The minimum shown on form 10 is regulated by consumption and the length of time required to replenish stocks with new material. The warehouse stocks governed by the purchasing department are handled in the following manner:

When the stock clerk posts the credits to the stock sheets and finds that a stock is approaching its minimum quantity, a metal signal tab is clipped onto the stock sheet in the lower right corner. The stock clerk then goes through the books monthly and requisitions the needed material. However, when a large quantity of a particular supply is drawn from stock and the total on hand is reduced to, or below, the minimum requirements the stock clerk immediately writes a requisition for supplies to replenish the item.

In the lower right corner of the stock sheet, a short column is headed "Ordered." This space is used by the stock clerk for noting in lead pencil his requisition number and quantity ordered, thus removing the signal from the sheet. This reference is left on the sheet until the quantity ordered has been received and posted on the sheet. It is then erased and the space is again ready for use the next time that material is ordered. By this method, when hurriedly perusing this sheet in the meantime, the clerk is informed that the material is on order.

Stock Sheet for Timber and Oil

Form 11 (fig. 15).- Form 11 is used for recording only stocks of timber and fuel oil. This larger size is used because the debit entries are so numerous - the commodities being received daily - that form 10 would not be suitable. The issues are posted daily on form 11 in the specially provided date columns appearing across the top of the form. The prices used in charging out the amount of timber and fuel oil are the average cost for the month.

Order on Storekeeper

Form 12 (fig. 16).- Form 12 is used for ordering supplies from stock and is signed only by authorized persons in the various departments. These supplies are credited daily on form 10, and are charged daily on the Supplies Issued sheet, form 13, after which they are filed numerically, according to account charged. When a department is interested in knowing the total charges against a particular account, form 12 is made in duplicate, the copy being sent to the department ordering, after prices and extensions have been shown thereon. In the exhibit it will be noted that 12 of the shovels ordered have been issued from stock.

Supplies Issued Sheet

Form 13 (fig. 17).- Form 12 is posted on the Supplies Issued sheets, form 13, which, at the close of each month, are sent to the accounting office to be entered in the general books of account. The left column on this sheet headed "Invoice Number" is used for entering the number of the invoice covering special items, which are purchased for specific purposes and delivered directly to the point of consumption, and are not entered on warehouse stock records. This class of material is charged out on form 12, the same as warehouse material, but this special plan makes it unnecessary to enter such material on the stock sheets.

One sheet is used monthly for each account number to which supplies have been charged. From the exhibit it is noted that the 12 shovels issued have been charged to account 19, which represents the 720-foot haulage level, and the classification credited in "Tools." The headings of the various ruled columns are for the 24 classifications of supplies.

Record of Supplies Used

Form 14 (fig. 18).- In studying form 13 the question might arise as to where and when the accounting department obtains the total monthly amount to be credited to the various

Location__________

Account__________ Price Unit__________ Reorder at__________ Article__________

	Date	Order	Quantity	Price	Amount	Acct.	Month	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	Total Used	Price	Amount	

Form 11

Figure 15.- Stock sheet for lumber and oil

MIAMI COPPER COMPANY

ORDER ON STOREKEEPER

12 - 23 1931

To Storekeeper:—

Please deliver to bearer the following material:

Charge Account No. 19

To be used for: 720' level haulage drifts

Quantity	ARTICLES	Class		Price		Amount	
12	#4 L.H. R.P. Miners shovels.	Tools			$3.33	10	00

This order must show:
Department where material is used
Job No. or Shop No.
Exact nature of work material is to be used for

L. G. Hubbs.

Form 12

Figure 16.- Order on storekeeper, in duplicate

MIAMI COPPER COMPANY
MIAMI, ARIZONA

SUPPLIES ISSUED

ACCOUNT 720' Haulage Level   MONTH DEC 1931   ACCOUNT NO. 19

INV NO	DAY		Tools										TOTAL TO DATE	DAY
	23	12-#4 LHRP Shovels	1000										1000	23
	TOTAL													

Form 13

Figure 17.- Supplies issued sheet

Form 14
Figure 18.- Record of supplies used

STATEMENT NUMBER

TO Brown Shovel Company,
Chicago, Illinois.

ITEMS	AMOUNT	ITEMS	AMOUNT
Inv 12/1/31 (40922)	500 00		
Less 2%	10 00		
	490 00		

NO RECEIPT NECESSARY

IF INCORRECT DO NOT USE CHECK BUT RETURN BOTH STATEMENT AND CHECK FOR CORRECTION
OTHERWISE DETACH CHECK FOR PAYMENT AND RETAIN STATEMENT

MIAMI COPPER COMPANY NUMBER

MIAMI, ARIZ., Dec. 8, 1931

PAY TO THE ORDER OF Brown Shovel Company $490.00

IN FULL SETTLEMENT OF ACCOUNT AS SHOWN ON ACCOMPANYING STATEMENT

TO CENTRAL HANOVER BANK
& TRUST CO.
NEW YORK, N. Y

SAMPLE COPY
GENERAL MANAGER

SAMPLE COPY
CHIEF CLERK

Brown Shovel Company,
Chicago, Illinois.

Inv 12/1/31 (40922)	500 00
Less 2%	10 00
	490 00

CORRECT

PURCHASING AGENT

EXAMINED AND ENTERED FOR PAYMENT

CHIEF CLERK

Dec. 8, 1931

Brown Shovel Company $490.00

Form 15

Figure 19.- Voucher check and statement, made in triplicate

supply classification accounts carried in their general ledger. At this point is exhibited form 14 which, however, is not a part of the supply department's system. It is kept by the accounting department for the purpose of summarizing the credits mentioned from form 13, Supplies Used. The totals of the classification columns on form 14 are credited by the accounting department to the respective classification accounts in the general ledger. Since only the totals are posted monthly it is apparent that the use of this form eliminates many individual credit postings to the supply classification accounts.

Voucher Check and Statement

Form 15 (fig. 19).- The Voucher Check (form 15) is partially discussed in the preamble of this article, but it might be well to state that the original copy contains a detachable statement showing the payee's invoice number, date, discount taken, freight deductions made, and in fact a complete résumé of the account. This of course is mailed with the check portion, which is detached from the statement by the payee before depositing the check in bank. On this statement is also shown (in brackets) the number assigned to the invoice upon its receipt. This makes a ready reference when checking an account, because, as previously explained, all invoices are filed numerically according to the consecutive number assigned upon their receipt. In the exhibit it will be noted that the 50 dozen shovels purchased on form 3 have now been paid for.

Physical Inventory Form

Form 16 (fig. 20).- This form is used for recording the physical inventory. The items are listed from the stock books, and the sheet is passed to the warehouse employees for recording quantities. They show the date the count is made, and the location or stock number, and the quantity, in their respective columns. This sheet is immediately passed to the supply office and reconciled with the stock ledger. An example of inventory of the shovels purchased has been entered in the exhibit.

Detailed Inventory Form

Form 17 (fig. 21).- This form is used for submitting the inventory at the end of each fiscal year and is written up in triplicate. The original is sent to the New York office, the duplicate copy is delivered to the Miami accounting office, and the triplicate copy is retained in the purchasing department office.

The procedure of writing up this inventory is as follows: At the end of the inventory period — December 31 — the description of the material, its classification, and article number are transferred from the stock ledger to the inventory sheets as shown in the exhibit. Then on January 1, after form 12, Orders on Storekeeper for December 31 stock withdrawals have been posted to form 10, the quantity of supplies on hand and the unit price, as shown on form 10 is entered by one of the clerks who signs his initials on the line marked "Entered by."

These entries are then called back by another clerk, who initials on the line "Called by." The clerk who checks the called figures on the inventory sheet signs his initials on the line "Checked by." The next step is the extension, which is made by a clerk who initials on the line "Extended by." These extensions are then checked by a clerk who initials on the line "Examined by."

After the entire inventory is written, the pages are numbered consecutively in the upper right corner, and recapitulations are made of the 24 supply classifications. By this method the entire inventory of about 8,000 items may be completed in four or five days.

Price Record Card

Form 18 (fig. 22).- Form 18 is a record of prices paid for materials. It is posted from the invoice just before placing it in its permanent file. This record, however, does not show every purchase. Entries are made on it only when price changes occur, whether up or down. These price record cards are filed alphabetically, under name of the article, and serve as a ready reference of actual costs.

FILING SYSTEMS

Specification and Parts Lists File

This file contains specifications and parts lists of machinery and equipment. These lists are filed alphabetically under the name of the article, in an upright filing cabinet using 9 by 12 inch folders.

Price File.- A price file is also maintained, using an upright filing cabinet for 9 by 12 inch folders. Contracts and current price lists, with discounts, are also kept in an upright filing cabinet taking 9 by 12 inch folders. These are arranged alphabetically according to the name of the article. Alphabetical division cards are used in the cabinet drawer. Contracts have separate folders, with the name of the commodity written on them. The more important commodities have separate folders, and the less important ones are placed in folders under the letter designating their name.

Correspondence File

The correspondence is filed alphabetically according to subject, in an upright filing cabinet using 9 by 12 inch folders. In this system there are established the following divisions and master numbers, and each subject under a particular division is given a subnumber. For instance, under the heading of "Electrical Supplies" which is represented by master number 9, the designation for motors is 9-1; for transformers, 9-2; etc.

Master No.

1 - Miscellaneous mine and milling machinery
2 - Miscellaneous supplies
3 - Iron and steel articles
4 - Timber and lumber
5 - Drills and parts
6 - Automobiles and trucks
7 - Building material
8 - Power-plant equipment
9 - Electrical supplies
10 - Oils and greases
20 - Railroad matters
21 - Accounts
22 - Miscellaneous matters

Catalog File

Catalogs are filed in built-in shelves, and are grouped under the following general headings:-

Number
1 - Electrical supplies
2 - General hardware
3 - Instruments
4 - Iron and steel
5 - Machinery
6 - Miscellaneous
7 - Paint, glass, and wall coverings
8 - Pipe and fittings
9 - Power-plant equipment
10 - Rubber goods, packing, and insulation
11 - Stationery, and office equipment
12 - Tools
13 - Transmission and material-handling equipment.

The general headings are recorded on 3 by 5 inch cards. These headings are further subdivided under individual items. An example is taken from a card headed "Tools":

Number
12 - Tools
12 - 1 to 49 - Shovels
12 - 50 to 99 - Jacks
12 - 100 to 149 - Welding and cutting
12 - 150 to 199 - Abrasives
12 - 200 to 249 - Brushes and brooms
12 - 250 to 299 - Machine tools
12 - 300 to 349 - Wood-working machine tools
12 - 350 to 399 - Mechanic's tools

Numbers for 49 catalogs on shovels have been allowed. These catalogs are numbered 12-1, 12-2, etc., which permits room for expansion when new ones are received, thus keeping within the same group all catalogs covering certain articles. The catalogs are also alphabetically indexed on 3 by 5 inch card records under firm name.

FORMS USED

Figure	Form No.	Purpose	Actual size, inches
3	1	- Requisition on purchasing department	8½ x 7¾
4	2	- Inquiry form	8½ x 10¾
5	3	- Order form	8½ x 14
6	3A	- "B" Order form	8½ x 10¾
7	3B	- Requisition on New York purchasing department	8½ x 14
8	4	- Acknowledgment tracer	5½ x 3¼
9	5	- Tracer for shipment, or invoice	5½ x 3¼
10	6	- Receiving report	8¼ x 7
11	7	- Tracer for additional copies of invoice or shipping papers	4¼ x 5½
12	8	- Oil receiving report	8½ x 5½
13	9	- Employees' requisition for tools	8½ x 5½
14	10	- Stock ledger sheet	12 x 4½
15	11	- Stock sheet for timber and oil	17¾ x 16¾
16	12	- Order on storekeeper	7¾ x 4½
17	13	- Supplies issued sheet	16½ x 15¾
18	14	- Record of supplies used	38 x 16½
19	15	- Voucher check and statement	8¼ x 8
	15A	- Duplicate of voucher	8¼ x 8
	15B	- Triplicate of voucher	8¼ x 8
20	16	- Physical inventory form	8½ x 11
21	17	- Detailed inventory form	9½ x 12
22	18	- Price record card	8 x 5

1931 Inventory
Tools

DATE	STOCK NUMBER	QUANTITY	✓	MATERIAL
12/28	2	588	✓	#4 LHRP Miners Shovels

Form 223 1m 4-30 MP&PCo B9718

Form 16

Figure 20.- Physical inventory blank

MIAMI COPPER COMPANY

INVENTORY

Sheet No. 70

Page

Entered by EM

Called by MQ

Checked by CGE

Extended by EH

Examined by CGE

Tools

	QUANTITY	DESCRIPTION	PRICE	EXTENSION
	588	#4 LHRP Miners Shovels	833	1000

Form 17

Figure 21.- Detailed inventory blank.

Shovels

DESCRIPTION	VENDOR	ORDER	INV DATE	F. O. B	UNIT	INV COST	TRANS.
#4 Miners L.H., R.P.	Brown Shovel Co	11772	12/1/31	Miami	Doz	$10.00	Paid

Form 18

Figure 22.- Price record card

I. C. 6624 June. 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MAGNETIC CONCENTRATION METHODS AND COSTS OF WITHERBEE, SHERMAN & CO., MINEVILLE, N.Y.

BY

T. F. MYNERS

I.C. 6624
June, 1932

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MAGNETIC CONCENTRATION METHODS AND COSTS OF WITHERBEE, SHERMAN AND COMPANY, MINEVILLE, N.Y.[1]

By T. F. Myners[2]

INTRODUCTION

This paper, describing the concentration methods and costs of Witherbee, Sherman and Company at Mineville, N. Y., is one of the series of papers on milling methods and costs being prepared by the United States Bureau of Mines. It deals principally with the production of high-grade concentrate.

ACKNOWLEDGMENT

The author is indebted to C. F. Jackson, principal mining engineer, United States Bureau of Mines, to A. M. Cummings, general superintendent of Witherbee, Sherman and Company, and to the company's engineering staff, for assistance in preparing this paper.

MINING

The mines and mills are situated at Mineville, N. Y., about 6 miles northwest of Port Henry on Lake Champlain and 1,100 feet above it.

Three mines furnish the crude ore. The lower Old Bed has a comparatively rich crude ore, high in phosphorous content but low in silica. New Bed has a low-grade crude ore, low in phosphorus and high in silica. Harmony yields a crude ore midway between that of Old Bed and New Bed. These mines are at present served by two mills, No. 4 and No. 5, but their crude has usually been run through No. 4 mill.

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1. The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6624."

2. One of the consulting engineers, U. S. Bureau of Mines, and assistant superintendent, Witherbee, Sherman and Company.

The open-stope method with pillar support is employed in mining. Extensive use of mechanical loading devices results in the hoisting of large chunks and necessitates the use of primary crushers at all pit heads. There is no sorting of waste rock underground; all material broken goes through the mill. As a result, when the amount of rock development is large, the grade of ore treated by the mill is adversely affected, although the vein itself continues uniform in grade.

TRANSPORTATION OF CRUDE TO MILLS

Old Bed ore is drawn from two bins; one is of circular, flat-bottomed steel construction with a capacity of 500 tons; the other of square, flat-bottomed concrete construction with a capacity of 1,000 tons, discharging into a 50-ton self-propelled lorry car driven by a 52 hp., 25-cycle motor. The lorry car carries the ore 400 feet to a receiving hopper beneath the track at No. 5 mill, whence it is fed to the crude-ore conveyor.

Harmony ore as indicated in the flow sheet, figure 1, after being discharged to a 36-inch belt conveyor, is loaded into 50-ton steel railroad cars and is hauled by the L. C. & M. Railroad to a feed track at No. 5 mill, whence the cars are dropped to the receiving hopper just mentioned.

When Harmony ore is being treated in No. 4 mill, the cars are placed over a raise driven to the surface from the top of a slope pocket in the New Bed mine. The ore is drawn from the pocket to trolley cars, hauled to a storage pocket at the main slope, hoisted to the surface, and then follows the course of New Bed ore as shown on No. 4 mill flow sheet (fig. 2).

CONSTRUCTION

The mill buildings are tall, relatively narrow, and are of steel and concrete construction with corrugated, galvanized-iron sides and roofs, or with concrete-block sides and corrugated-iron roofs. The crushing machines are so intimately connected with the separating department that, with the exception of the primary and secondary crushers, they are located directly under the magnetic separators. The mills are locally known as separators. An all-dry process is used, with electromagnets for effecting separation.

CAPACITY OF MILLS

As originally designed, No. 5 mill, treating Old Bed ore, had an hourly capacity of 150 tons of crude; No. 4 mill, treating New Bed ore, of 125 tons of crude; and No. 3 mill, treating Harmony ore, of 175 tons of crude. No. 3 mill burned in 1923, and since then Harmony ore has been treated in either No. 4 or No. 5 mill, as conditions warranted. Neither mill handles Harmony ore at the same rate that No. 3 mill did.

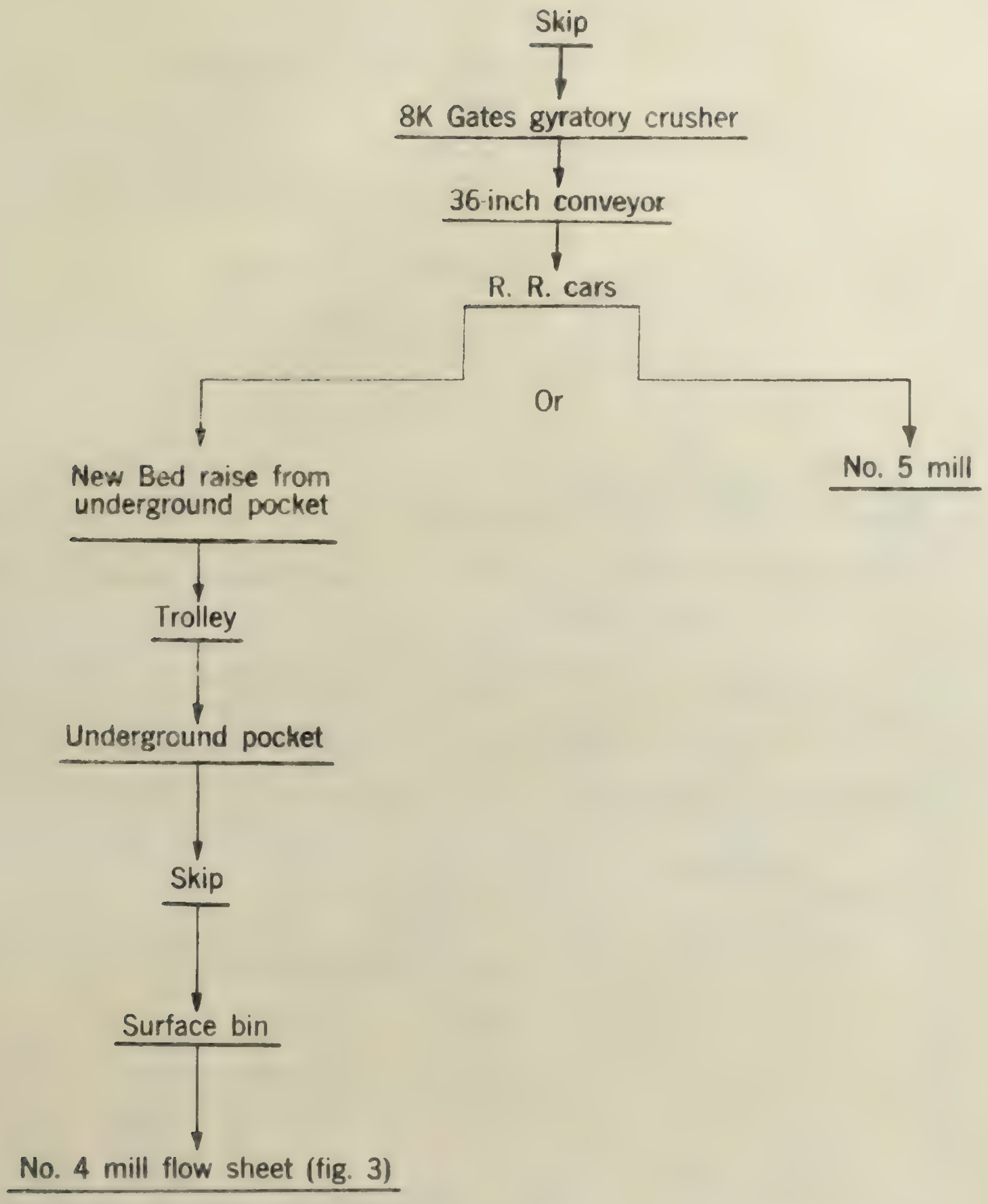

Figure 1.—Flow sheet, Harmony ore

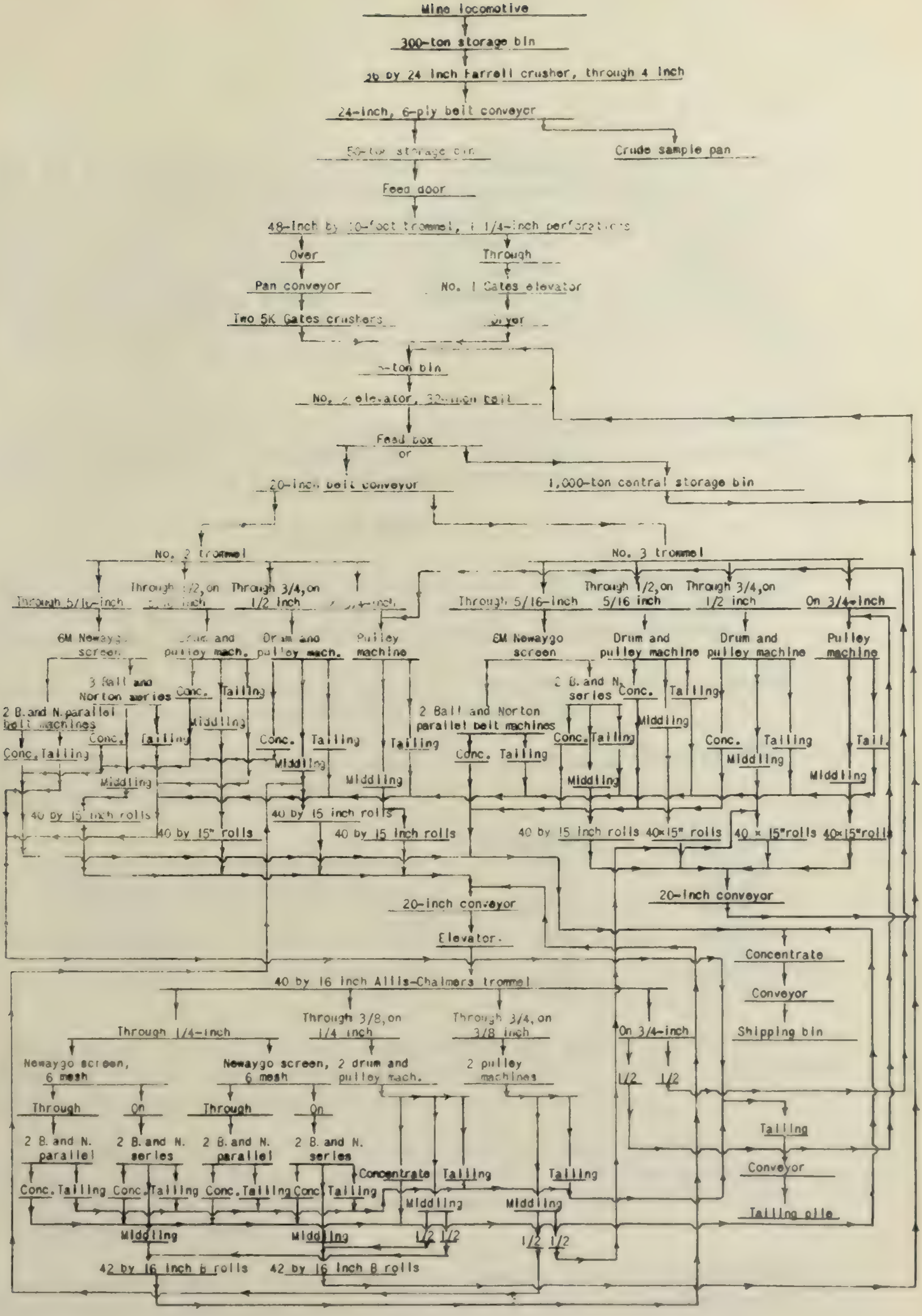

Figure 2.- Flow sheet of mill No. 4

Due to the demand for a high-grade concentrate - 68 to 70 per cent of iron, with a correspondingly decreased phosphorous content - the present capacity of each mill is approximately 60 per cent of that originally obtained. With the addition of greater roll capacity, and with more magnetic separators, it is hoped to restore the original capacity of the different units. In the design for rebuilding No. 3 mill, much thought is being given to the present demands for a high-grade product - that is, a concentrate carrying 68 per cent or more of iron.

The mills are located on level portions of ground, near the mine each serves. Because of the heavy return middling load in the mill, nothing is gained by the use of terraced construction.

POWER

Power for this property is purchased from the New York Power and Light Corporation. It is delivered to the mills at 3,300 volts and is then transformed to 440 volts for the 3-phase, 25-cycle motors.

Direct current at 125 volts for actuating the separator magnets is furnished by a motor generator set at each mill.

No. 5 mill is equipped with motors having a total capacity of 860 hp. This is divided as follows:

	Horsepower	Per cent
Crushing and grinding.........	220	25.58
Magnetic separators...........	100	11.63
Conveying.....................	285	33.14
Elevating.....................	200	23.26
Screening.....................	55	6.39
Total..	860	100.00

In addition, there is a motor-generator set giving an output of 50 kilowatts at 125 volts. It is driven by a 75-hp. motor. All motors are of the 3-phase, 25-cycle, 440-volt, induction type.

Conveying accounts for the largest single item of power consumption. Crushing is the next largest, and elevating is third. The transportation of materials accounts for 56.40 per cent of the total power used at the mill.

No. 4 mill, because of the lean ore treated by it, and the resulting heavier circulating load, will show a greater percentage of power used in the transportation of material.

ORE TREATED

The feed to the mills is magnetite associated with gneiss and apatite. The gneiss is of two types: dark or basic, characterized by a predominance of hornblende and biotite and absence of free quartz; and the light-colored or acid variety, characterized by freedom from biotite and hornblende and predominance of quartz. The crude from the Old Bed is nearly all "pure ore" and clean rock; from the Harmony, pure ore, lean ore (in which the magnetite grains are disseminated throughout the light gneiss), and rock; from the New Bed, mainly lean ore. The pure ore is composed of pure crystals of magnetite mixed to a greater or less extent with grains of apatite, in which form the phosphorus occurs. The lean ore varies from coarse crystals heavily sprinkled throughout the light-colored, acid gneiss of the hanging wall to small crystals rather sparingly sprinkled through the gneiss. The degree of concentration of the magnetite crystals in the gneiss seems to vary with the distance the material lies from the pure ore, the more heavily sprinkled ore lying nearest the vein.

Martite, a hematite with crystallization and appearance similar to magnetite, is also present in the ores.[3] It is, however, not responsive to magnetic attraction of the intensity employed at Mineville. The feed to the mills in 1930 was as follows:

	No. 5 Mill Fe, per cent	No. 4 Mill Fe, per cent
Old Bed..................	52.57	--
Harmony..................	--	40.26
New Bed..................	--	29.79

MINERAL CHARACTERISTICS

There is great variation in crystal size. Some crystals measuring 1-1/4 inches along an edge have been found, others are smaller than the head of a pin. In general, the rich ore breaks finer than lean ore or rock, and the mill foreman uses the amount of fines present in the crude ore as a rough indication of its grade.

The moisture content of the crude ore is approximately 3 per cent. This is reduced to 1/4 or 1/2 per cent before separation. Rich ore is apt to be granular. Apatite occurs as grains filling the spaces between magnetite crystals.

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3. Norton, S., and LeFevre, S., The Magnetic Concentration of Low-Grade Iron Ores; Trans. Am. Inst. Min. and Met. Eng., vol. 56, 1917, pp. 892-916.

HISTORY OF MAGNETIC CONCENTRATION AT MINEVILLE

The first known use of magnetic concentration was shortly after the acquisition of the "21" mine in 1853 by the American Mineral Co. They introduced the process for the purpose of extracting the apatite, selling the magnetite to the founders of Witherbee, Sherman & Co. as a by-product. The process was not a success, however, and was finally abandoned.[4]

A period of wet-milling followed and several small units were built at various locations in the Mineville area for the purpose of effecting a separation by jigging. These suffered the same fate as the first magnetic separator and were abandoned.

Forty years ago the problem of magnetic concentration was again attacked by using the Wenstrom drum-type machine to effect a separation. No. 1 mill, the first dry magnetic concentrator to handle magnetite on a commercial scale, was built in 1902, and No. 2 mill was erected shortly afterward. Both of these mills treated Harmony and Old Bed ores. In 1910 No. 3 mill was built near Harmony A shaft to treat Harmony ore. No. 4 and No. 5 mills were then constructed to treat New Bed and Old Bed ores, respectively. Of the five mills erected only two, No. 4 and No. 5, are in use, the others having been destroyed by fire.

A new mill to replace No. 3 is being planned to make available a unit having a flow sheet adaptable to the treatment of Harmony ore and to release No. 4 mill for its original purpose of treating New Bed ore.

The dry magnetic concentration method of treatment is the result of experience at Mineville. Wet methods are not so satisfactory. They were tried during the experimental period previous to the erection of No. 1 mill, and from time to time wet units have been tested in combination with the present flow sheets. The last attempt was made during the years 1923-1925. This project was finally abandoned and efforts were directed toward the further refinement of dry-separating methods.

Table 1 shows the direct costs and concentration ratios since 1926. They represent the treatment of Old Bed ore by No. 5 mill since the resumption of operations in that year. They also represent the longest period of treatment of one ore by one mill during this period. Constant changes, necessary because Harmony ore has been run through No. 4 and No. 5 mills, often when other ores were being treated, have operated against obtaining the best all-round cost and efficiencies.

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4. The Iron Age, The Mineville Magnetite Mines: December 17, 1903.

Approximately 29 per cent of the increase in cost of milling during the 5-year period may be traced to separating, 10 per cent to crushing and grinding, 10 per cent to screening, and 14 per cent to elevating and conveying. The increased cost is due to the extra crushing and grinding and subsequent rehandling necessary for concentrating the ore to a richer product.

Table 1. - CONCENTRATOR COMPARISON

No. 5 Mill — Old Bed ore

Year	Crude tons	Fe per cent	Concentrate: Tons	Concentrate: Fe Per cent	Cost per ton of Concentrate	Recovery, per cent	Ratio of Concentration
1930	390,019	52.57	279,343	67.49	$0.437	95.59	1.396:1
1929	343,038	52.38	258.173	66.29	.342	95.25	1.329:1
1928	340,563	54.60	277,083	65.72	.276	97.93	1.229:1
1927	283,706	52.56	224,623	64.10	.325	96.56	1.263:1
1926	300,460	54.03	245,466	64.42	.238	97.41	1.224:1

Note: The long ton of 2,240 pounds is used as the basis of all tabulations.

BREAKING

The primary crusher for each mill is placed at the pit head of the mine serving that mill. At the Old Bed mine there is a 30 by 18 inch Blake jaw crusher, at the Harmony mine an 8K Gates gyratory crusher, and at the New Bed a 36 by 24 inch Farrell-Bacon jaw crusher. A grizzly is used ahead of the primary crusher in only one instance - at the Old Bed mine where ore is dumped from the skip on an inclined, stationary grizzly with openings set at approximately 2-1/2 inches when new. The other two crushers receive run-of-mine material, the feed to the 8K crusher being dumped directly into the bowl and the jaw crusher being fed from a 300-ton storage bin. All primary crushers are set to deliver a 4-inch product. Figure 3 is a flow sheet of No. 5 mill, and Figure 2 a flow sheet of No. 4 mill.

The maximum size of pieces delivered to the crushers is limited only by the pocket-gate opening, which is 4 feet square. Pieces 2 feet in largest dimension are common at the Old Bed and Harmony mines, where mechanical mucking is practiced. At the New Bed mine these sizes are not common because of the prevalence of hand mucking. As the program of mechanizing the mine progresses, the proportion of large pieces to small is increasing, as is also the average size of the larger pieces.

Figure 3.- Flow sheet of mill No. 5

Approximately 20 per cent of the ore is undersize. The undersize is removed by the grizzly at the Old Bed mine but goes through the crushers at the other two. Each unit has a capacity of 100 tons per hour when delivering 4 inch material. In each case the crusher is belt driven.

INTERMEDIATE CRUSHING

Each mill has two 5K Gates gyratory crushers taking 4 inch material from the primary units and delivering it as 2 inch material. Those at No. 4 mill have a capacity of 100 tons an hour working on lean ore, while those at No. 5 mill have a capacity of 150 tons an hour, working on rich ore. Of this amount, 100 tons is crude ore fed to the mill and 50 tons is oversize returned from the first screening operation. In the latter installation the crushers are in closed circuit with the screen. In the new No. 3 mill it is proposed to use 6K Gates gyratory crushers having a capacity of 100 tons an hour, working on the Harmony crude ore.

At No. 5 mill a gravity screen having 2 inch round holes is placed ahead of the intermediate crusher. The fine material is by-passed and joins the crusher product in the elevator to the trommel previously mentioned.

At No. 4 mill a trommel having 1-1/4 inch perforations is placed ahead of the crushers. The oversize from it goes to the crushers while the undersize is elevated directly to the dryer.

All intermediate crushers are belt driven from a line shaft.

Wearing parts, wherever possible, are of manganese steel. The only exceptions are the brass wearing rings and the cast-steel bevel wheel. Cast-iron bevel wheels are made at the company's foundry but have not proved so satisfactory as the cast-steel wheels and are installed now only as emergency units. An extra top shell is provided at each mill. When a set of concaves is worn down, the new set is placed in the spare shell so that when the change is made, the whole concave assembly is removed and replaced by the spare set, the old concaves being removed from the shell when mechanics can be spared from other repair work. Shortly before it is necessary to change concaves again, the correct diameter of the discharge opening is determined and the spare set is arranged for that condition. Concaves are in two halves, upper and lower. When it is necessary to draw in the concaves to compensate for wear, only the lower set is changed. One set of upper concaves will outwear two of the lower. A like operation is followed when it is necessary to change mantles. At Mineville a new mantle is run with an old set of concaves, or vice versa, to obtain what is felt to be the best combination of crushing efficiency and economy.

In this way, two runs are obtained from each part without sacrificing efficiency. Manganese-steel mantles (corrugated), are used which slip over and lock on cast-iron heads permanently fastened to the shafts. When the crushing efficiency of the mantle is gone, it is replaced with a new one, already assembled on a spare shaft and head. With this practice, it is possible to maintain a high degree of crushing efficiency with but half the shutdown time or overtime work chargeable to crusher repairs.

The crushers are run at 350 r. p. m. (at the countershaft).

SCREENING

Screening practice at Mineville is not rigid, changes being made from time to time as the character of the feed to the product warrants.

At No. 4 mill a trommel precedes the 5K crushers. It is 48 inches in diameter, 10 feet long, and has 1-1/4 inch perforations. At No. 5 mill, because of space limitations, a gravity screen with 2 inch round holes performs the same duty. At this mill, a gravity screen having 3/4 by 1-1/2 inch slotted holes and a 48 inch by 10 foot trommel with 1-1/4 inch perforations for 5 feet, and 2 inch perforations for the remaining 5 feet, follow the crushers, being in closed circuit with them.

The gravity screen removes the fines from the feed, whence they are passed over a Ball and Norton drum separator. The concentrate picked by this separator goes directly to the shipping bin, and the reject or middling goes to the dryer. However, when making high-grade concentrate, this separator is not used, since much of its product approaches 3/4 inch. This size, known as "buttons," contains too much phosphorus to go into the shipping product. Even on finer material the phosphorous content of its product is apt to be high, since the material fed to it is damp and therefore not susceptible to clean separation.

The oversize from the gravity screen passes to No. 1 trommel, mentioned before. Here three sizes are made. The first half of the screen has 1-1/4 inch perforations. Through 1-1/4 inch material goes to the dryer with the undersize from the gravity screen. On 1-1/4 inch and through 2 inch material goes to the pulley-type separators where tailing and middling products are made. The plus 2 inch material returns to the 5K crushers.

From the rolls, the crushed products join the dried ore from the dryer. The combined products go to No. 2 and No. 3 trommels, which are 48 inches by 16 feet in size; the first 12 feet being jacketed with a 72 inch diameter jacket having either 1/4 or 5/16 inch holes, depending on the shipping product desired. Present practice at No. 5 mill calls for the use of 1/4 inch perforations in the first 8 feet of the jacket and 5/16 inch in the next 4 feet.

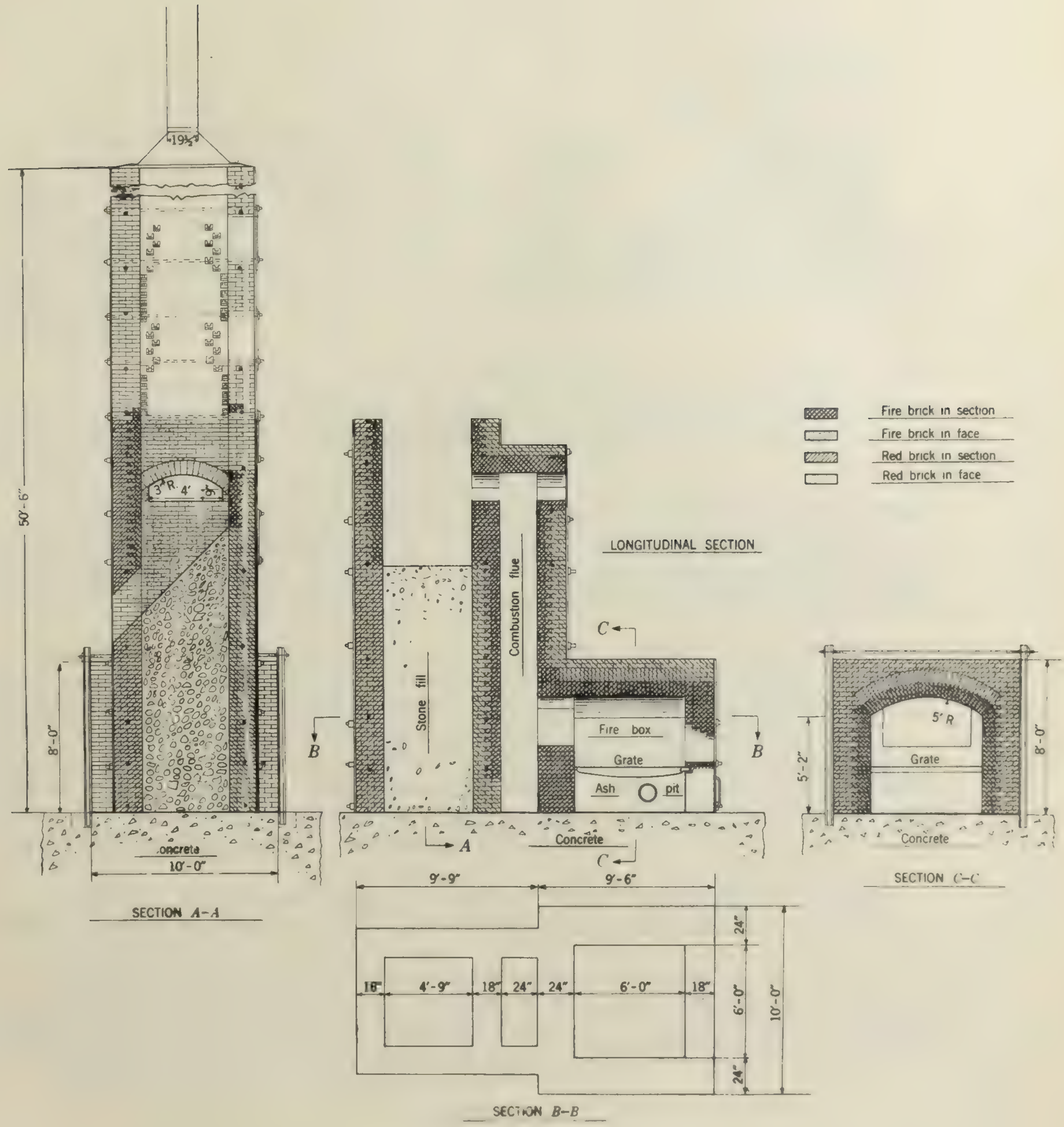

Figure 4.—Diagram of brickwork, No. 4 and No. 5 mill dryers

Minus 1/4 inch product goes to the belt machines; minus 5/16 inch goes to drum and pulley type combination machines. Plus 5/16 minus 3/4 inch material goes to pulley-type machines for separation into middling and tailing products, when making our present high-grade concentrate. Formerly they passed first over drums for a concentrate-middling separation and then to the pulley-type machines. Plus 3/4 minus 2 inch material passes over a pulley-type machine for a middling-tailing separation.

No screening below the 1/4 inch size is done at No. 5 mill. The necessity for making a high-grade concentrate requires a light current on the separators which automatically takes care of the screening problem.

At No. 4 mill, two 48 inch by 24 foot trommel screens divide the product, as is the practice at No. 5 mill. Here, however, four 8 by 5 foot Hum-mer screens, working on minus 1/4 inch material, make plus and minus 6-mesh products for the Ball and Norton belt-type separators.

Screens are provided for making plus and minus 10-mesh products for feed to the machines that effect the final separation before the material goes to the shipping bin. The oversize returns to the rolls while the undersize goes to the separators. Two finer 8 by 5 foot Hum-mer screens further divide the minus 10-mesh material before it is fed to the separators. Their main function, however, is to distribute the feed, and more attention is given at this point to feed distribution than to obtaining maximum screening efficiency.

Under present conditions it does not seem necessary to screen below 10 mesh in the separating section of the flow sheet.

Experiments are being conducted at present relative to the segregation of the phosphorous content of the concentrate in the various sizes of particles. It may prove advantageous to divide the concentrate by means of screens to make high and low phosphorous shipping products.

DRYER

The dryer is an essential part of the dry magnetic concentrator flow sheet. Unless the ore is bone dry, clean separation into concentrate and tailing is not possible.

The dryers used by Witherbee, Sherman & Co. are of the tower type, Rowand design (fig. 4). They consist essentially of a fire box and chimney. Feed is introduced at the top of the chimney or stack and falls against rising hot gases, the driest part of the ore stream being near the hottest gases at the point where they enter the stack.

A series of grids composed of T-section cast-iron bars set at right angles with each other, serves to impede the fall of material and to break up and distribute the ribbon of ore, exposing a greater drying area to the ascending gases. These sections fit loosely into racks set in the stack walls for ease in removal. They are reached for inspection and repair through doors set at intervals along the stack.

A combustion chamber, locally known as the flue, is between the fire box and the stack. Here an opportunity is afforded for the complete combustion of the coal gas. The flue is also a safety feature in the event that the discharge opening blocks, since the chamber must fill to the height of the bridge wall before the fire can be covered by the ore. Cleaning of this chamber is facilitated by a clean-out door at ground level.

Fire brick is used for the lining and arch of the fire box and flue and for the lining of the stack for several feet above the opening above the flue and the stack. This is done not only for protection against heat, but also to resist abrasion of the falling ore. Former practice was to use two courses of fire brick in the arches. At present, one course of wedge-shaped bricks 13 inches long, set on end, forms the arch. Arches so constructed are giving better service than those of two-course construction. Dryers are shut down for repairs to the brick-work about once a year under average conditions. Repairs were necessary about eight months apart with the old practice.

Normally, natural draft is relied on, but forced draft is provided for should it become necessary because of wet ore, unusually heavy feed to the dryer, or atmospheric conditions.

Feed to the dryer consists of minus 1-1/4 inch material. The sizes from 3/4 to 1-1/4 inch serve to prevent "blocks" by eliminating the bridging of fine ore on the grid bars.

Moisture content of the feed is approximately 3 per cent; that of the discharge is 1/4 to 1/2 per cent. Coal consumption is 1-3/4 tons of soft run-of-mine coal per day of 24 hours, or 0.073 tons per hour. The hourly consumption will be a trifle higher on one shift operation because of the necessity of keeping the dryer warm between shifts.

MAGNETIC SEPARATORS

Three types of magnetic separators are used in the Mineville district - pulley, drum, and belt. At Mineville the term "heads" is applied to the concentrate product, and it is so employed in the following description of concentration practice.

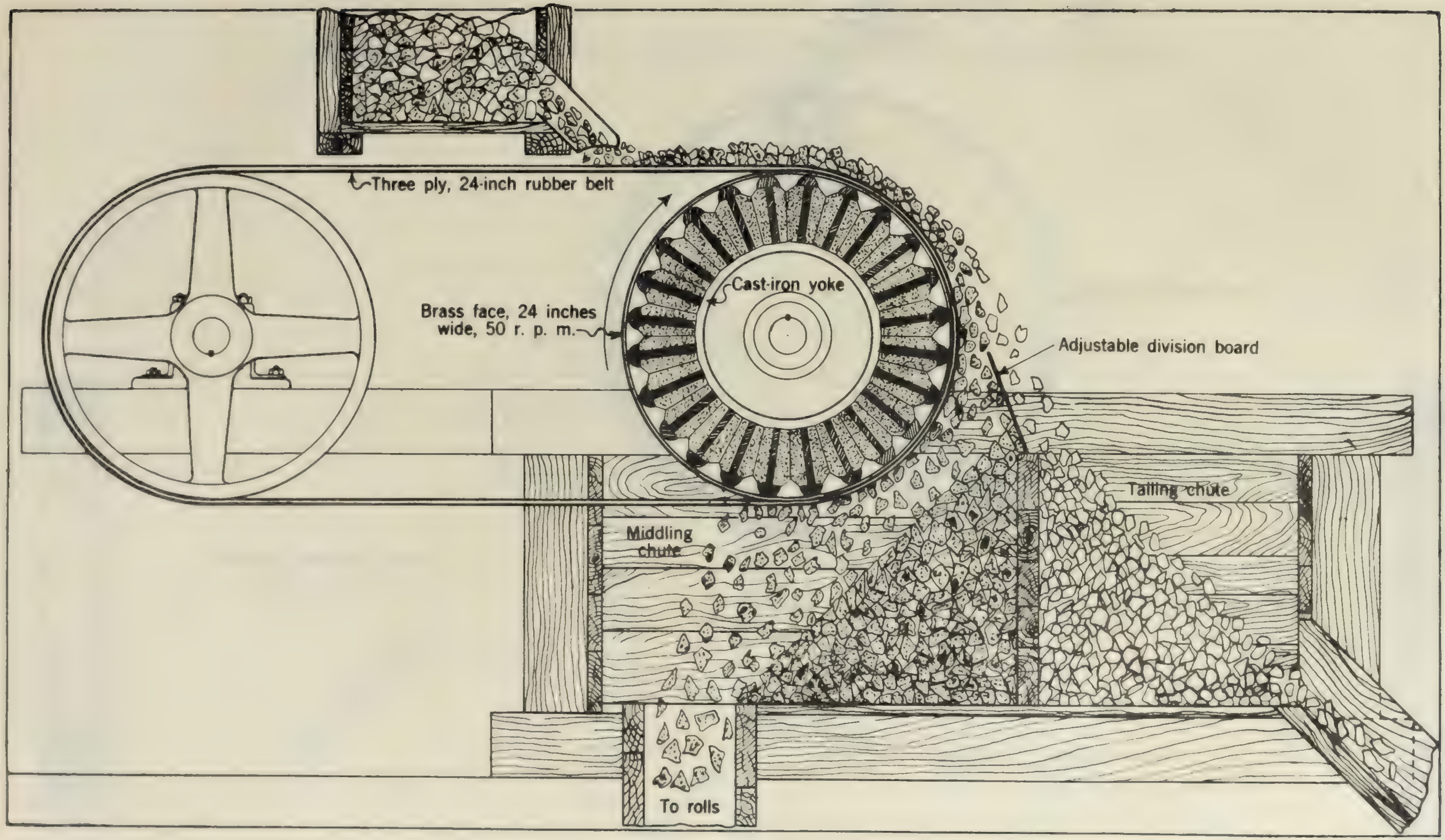

Figure 5.–Center section through the pulley-type separator, showing operation on a sized feed

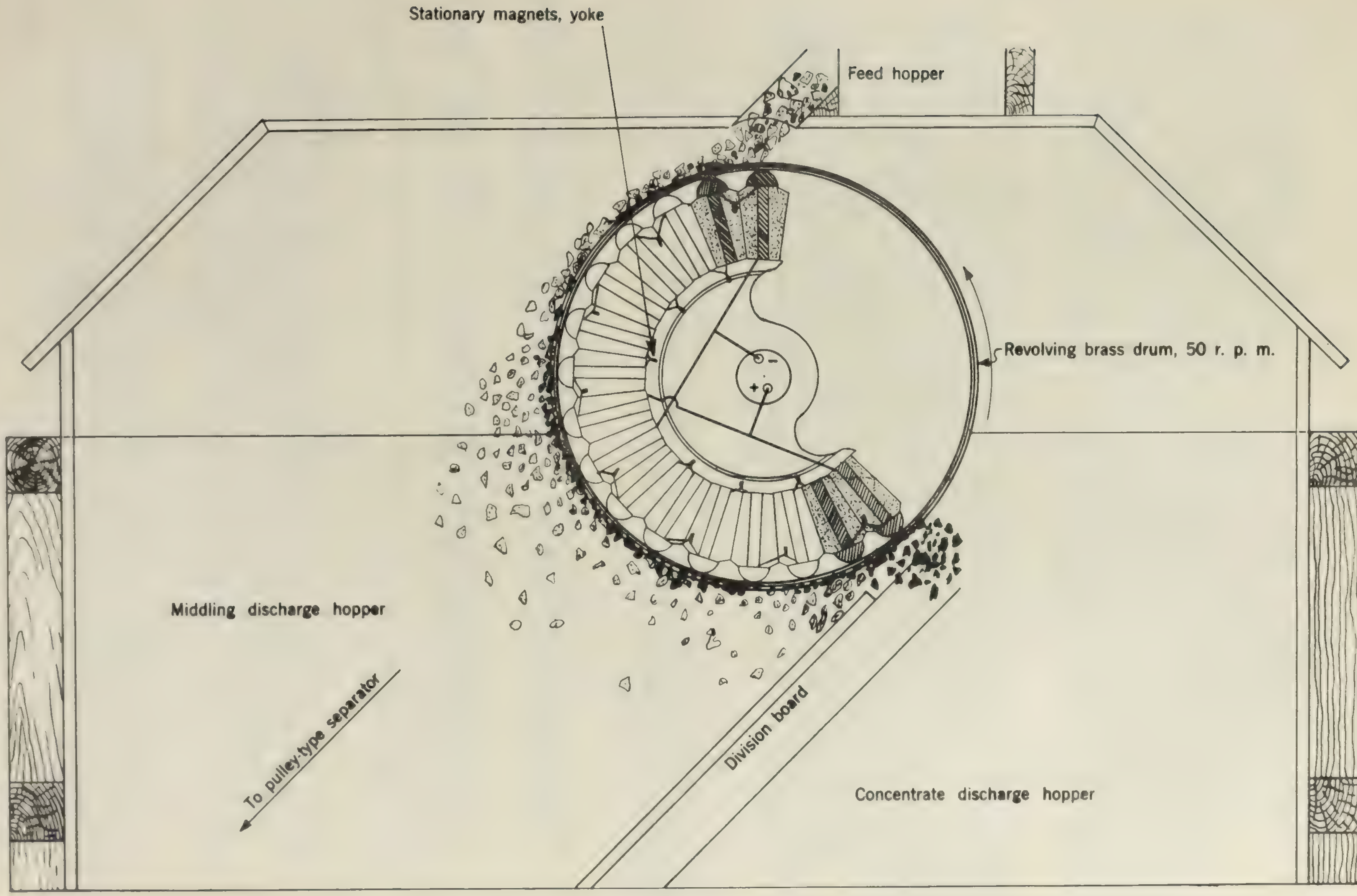

Figure 6.– Center section through drum machine working on sized feed

Pulley type.- The pulley type (fig. 5) of magnetic separator, as its name implies, forms the head pulley of a short conveyor. It consists of a group of magnets mounted in a full circle on an iron core, revolving with the pulley. The magnets are enclosed by a brass shell and two brass heads, and they are so wound that adjacent poles are of opposite polarity, thus producing concentrated magnetic fields at the shell and belt. They are capable of carrying a heavy current and will hold anything carrying an appreciable amount of magnetite, throwing off a comparatively clean tailing. The unit is used, therefore, primarily as a cobbing machine to make tailing and middling.

Feed to the machine strikes the belt at a point back of the pulley. As it comes into the magnetic field, magnetite is held against the belt by magnetic attraction while barren rock or ore too lean to be held is thrown off by centrifugal force as in the ordinary belt conveyor. The retained ore passes around the periphery of the unit until the divergence of the belt from the shell causes it to pass out of the magnetic field, when it falls into the middling chute to go to the rolls for further crushing. The elimination of tailing in the early stages of milling materially reduces the load on the rolls and the mill as a whole. The effect on the rolls is especially important, since the rock is much harder and less friable than the ore.

At Mineville, separation begins at 2 inches. Pulley machines treat this and each successive size down to plus 1/4 minus 3/4 inch. Generally they work in conjunction with drum-type machines, treating the middling made by them.

Tailing will average 4.20 per cent Fe on the 2 inch material to 1.85 per cent Fe on the plus 1/4 inch size, where the pulley-type separators end their work.

Maximum current carried on these machines is 25 amperes at 125 volts. Under ordinary conditions not over 21 amperes will be flowing.

The belt speed is 350 feet per minute, and the capacity is 3 tons per hour.

Drum Type.- The drum-type machine (fig. 6) consists of a closed cylinder of brass revolving about stationary magnets, mounted, as in the pulley machine, on an iron core and covering approximately two-thirds of the periphery of the shell. The magnets are so wound that adjacent poles are of opposite polarity. Electrical connections are made through a hollow shaft.

Magnetite ore is fed onto the upper part of the drum in the direction of rotation. Magnetite is held to the surface of the drum as in the pulley-type machine, while nonmagnetic material is thrown off by centrifugal force. There is this important difference, however:

In the pulley-type machine, magnetic particles are held stationary with respect to the pole which attract them, whereas in the drum machine the particles are in constant motion with respect to the poles.

The speed of the drum is so adjusted that the time taken to pass from pole to pole is less than that necessary for particles of magnetite to lose their secondary magnetism. The result is that a winnowing action takes place which releases any gangue entrained in the ribbon of feed on the drum. This happens because during the passage of a particle from one pole to another, there is a tendency for it to lose its secondary magnetism and pass out of the magnetic field, together with any nonmagnetic or weakly magnetic particles present. At each pole the strongly magnetic particle will be pulled against the drum, while nonmagnetic or weakly magnetic particles are thrown off into the middling hopper. When the particle reaches the end of the segment covered by the magnets, it is thrown by centrifugal force into the concentrate hopper.

This machine draws a comparatively light current and is used to make concentrate and middling. While it is possible for this machine to make a concentrate having 68 per cent of iron, its best field is in the production of 62 to 64 per cent concentrate (this applies to Mineville practice). Its reject is a middling product carrying 45 per cent of iron. The middling is treated on the pulley-type machines for the purpose of removing tailing.

Drum machines at Mineville contain 16 magnets drawing from 4 to 7 amperes at 125 volts. From 0.5 to 0.75 hp. is required to drive the drums. The peripheral speed is from 390 to 440 f. p. m. The capacity is approximately 10 tons of concentrate an hour.

Modified Double-Drum Type.- During 1930, the drum machines were modified (fig. 7). This new type has the general dimensions of the pulley machine. The magnets are mounted as in the standard drum type, except that 14 are assembled on the core. The upper 6 magnets are so connected that they are stronger than the lower 8 (fig. 8). The purpose of the change in design is to eliminate the nonmagnetic material from treatment by the machine as soon as possible and reduce the possibility of carrying entrained rock over into the concentrate. Elimination of the rock leaves a thinner stream of ore for the low-tension magnets to work on, bringing the practice nearer the ideal of a stream only one particle thick. During test work in the laboratory, it was possible to make three products - concentrate, middling, and tailing - from a single drum. This is not attempted in the mill, however. Two drums are mounted, one above the other. The upper drum makes a concentrate and a middling product. The middling from the upper drum falls to the lower drum and there the separation into middling and tailing products takes place. Having a thin bed of material to work on, the low-tension magnets of the upper drum make a clean concentrate, while the lean middling fed to the lower drum makes

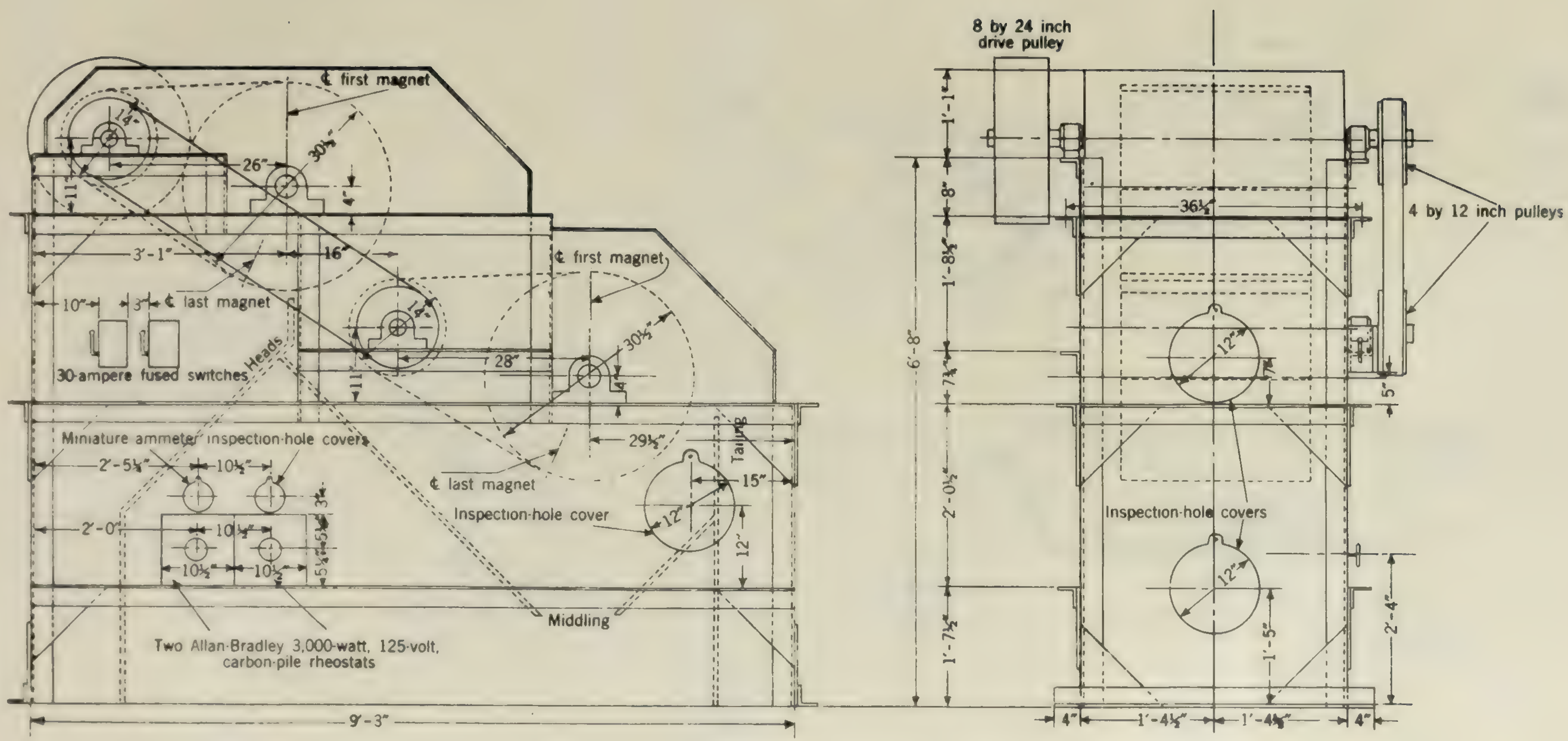

Figure 7.—Double-drum magnetic separator assembly

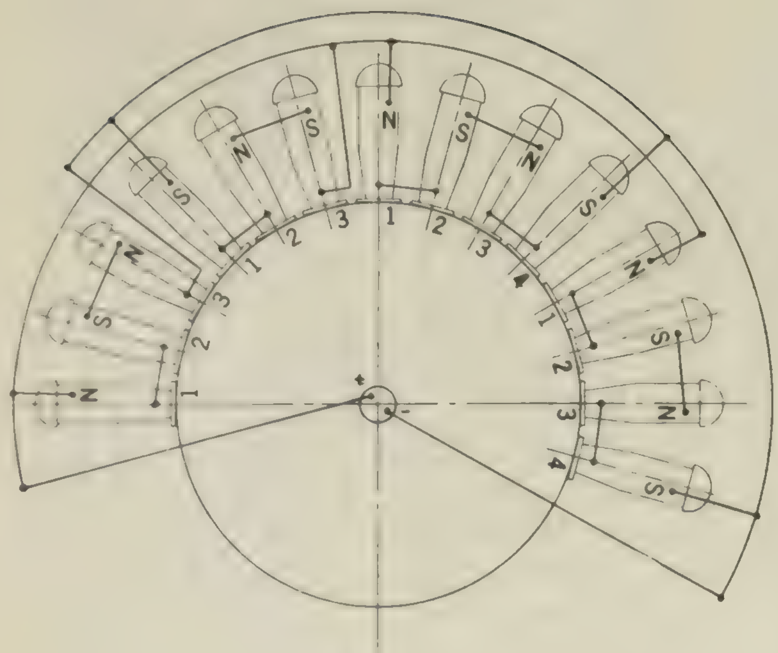

Figure 8 – Wiring diagram for special drum machine

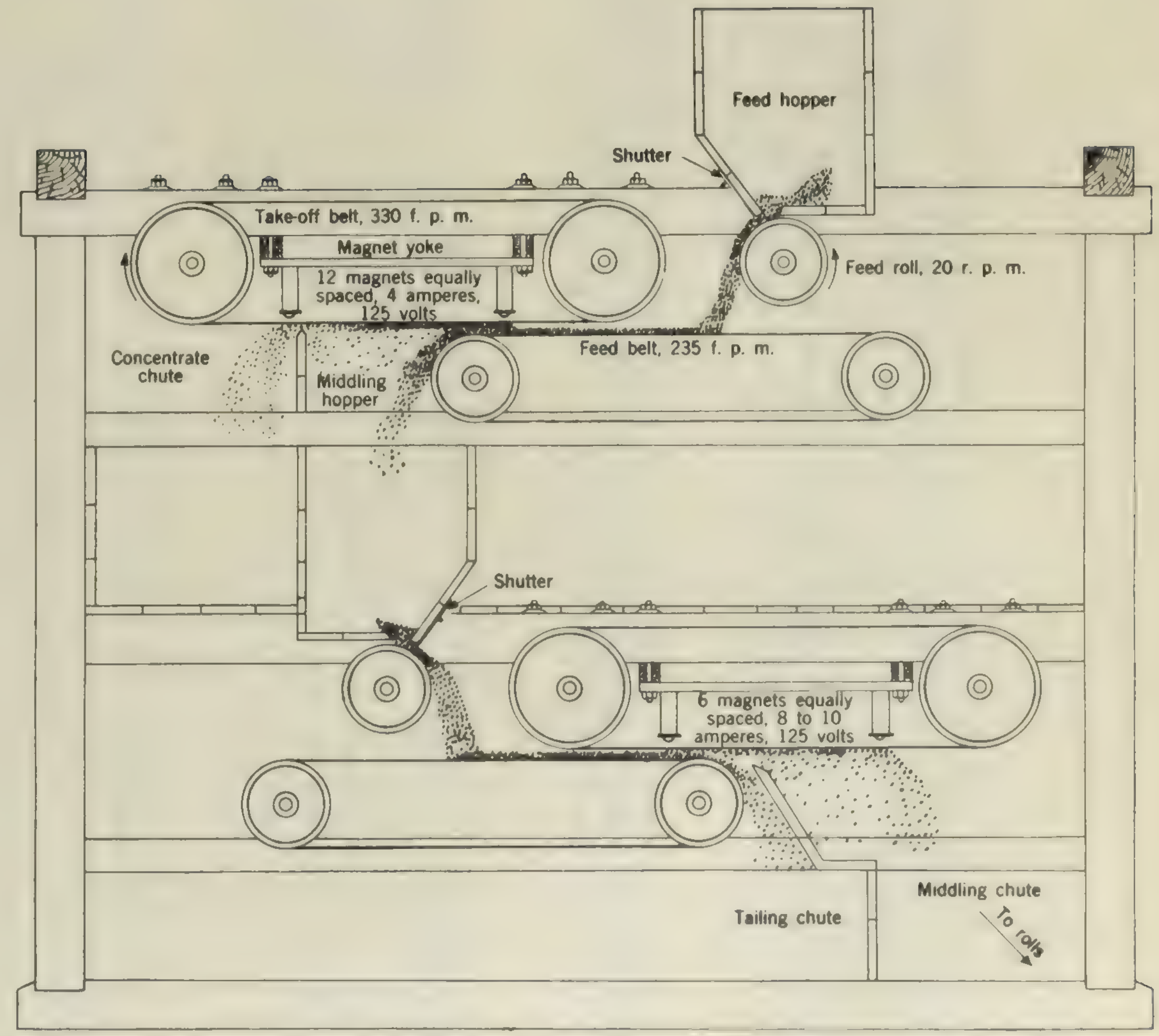

Figure 9.– Belt-type (series) machine, showing operation on fine-sized feed

a clean tailing possible. Working on mill ore, in its regular place in the flow sheet, this "double drum" machine makes 66 per cent Fe concentrate and 1.20 per cent Fe tailing. These are the results of magnetic determinations. As usually run in the mill, 14 magnets will draw 9 amperes at 125 volts on the upper drum making concentrate, while the lower drum making tailing draws 12 amperes. The peripheral speed of the drum is 340 f. p. m.

Rheostats for the regulation of the current carried by the machines are mounted within the frame, permitting accurate separating control. Such mounting is not practicable on our other types of machines.

Belt Type.- The belt-type machine (fig. 9) is especially adapted to working on the smaller sizes of ore. It consists essentially of a series of magnets in a horizontal position above a continuous belt known as the "pick-up" belt. The magnets are so connected that adjacent poles are of opposite polarity in order to introduce the agitating or sifting action necessary for efficient cleaning of the stream of ore presented to it.

Where, in the case of the drum machine the stream is held to the surface of the drum by magnetic force while the elimination of nonmagnetic particles is accomplished by centrifugal force, in the belt machine the magnets hold the magnetic particles against the force of gravity, the nonmagnetic or weakly magnetic middling pieces dropping out of the stream during the sifting action which takes place as the feed advances from pole to pole. The last magnet is placed over the division board to carry the cleaned product into the right hopper.

Treatment of the ore by belt machines begins at minus 1/4 inch size. At No. 4 mill the treating unit consists of two sets of magnets mounted one above the other with suitable arrangements for feeding each. At No. 5 mill the unit consists of three sets. Locally the unit is known as a "machine", so that we have 2-deck and 3-deck machines. These machines work in series. For instance, at No. 4 mill, feed is introduced from the storage bin to the top set of magnets where concentrate and middling are made. The middling from the upper magnets then passes to the lower set where middling and tailing are the products. The two sets of magnets carry different strengths of current. The upper has the weaker current and is the low-tension set, while the lower set, making the tailing product, has a strong current and is the high-tension set. The procedure is the same on the 3-deck machines at No. 5 mill, except that two sets of magnets are making concentrate and middling instead of the one set at No. 4 mill. This is due to the fact that the crude ore is richer at the latter mill and one set of magnets is not enough to pick out all the highly magnetic particles with a current weak enough to insure a clean product.

The upper belt is run at such speed that the particles are carried from one pole to the next before their secondary magnetism is lost - the same principle which applies in the drum-type machine. It is this principle which permits gravity to pull all but the highly magnetic particles out of the stream before the ribbon of feed passes out of the magnetic field into the concentrate hopper.

The belt machine works best on a sized product from minus 1/4 inch and 10 mesh; experience has proved that sizing is not important beyond this point, regulation of the current impressed on the magnets causing a selection of particle size.

The 2-deck machine is used in both No. 4 and No. 5 mills for working on minus 10-mesh material. Here the amount of iron present in the middling is not sufficient to pay for additional regrinding. The tailing from the belt machines working on this material is, therefore, relatively high in iron, but the amount is small and does not reflect adversely on the tailing of the mill as a whole.

Operating details of this machine are as follows:

Current........	amperes 6 to 12	
Belt speed.....	per minute feet	310
Capacity.......	per hour tons	16
The products are:		
Concentrate....	at 66 per cent iron tons	12.7
Middling.......		.9
Tailing........	at 2.0 " " " "	2.6

GRINDING

There is no fine grinding, as the term is ordinarily used, at the Mineville plant. The process is primarily one of roll-crushing, the finest stage being nominally 10 mesh.

Two sizes of rolls are in operation: 42 by 16 inch, style A, 42 by 16 inch, style B, and 40 by 15 inch. In general, the 42 by 16 inch rolls are used for the coarse stages and the 40 by 15 inch for the fine stages of grinding. All are belt driven from line shafts.

Minus 2 plus 1-1/4 inch middling from the pulley-type separators is the first material fed to the rolls. It goes to the 42 by 16 inch size. The product is 3/4 inch.

The next size treated by the rolls is minus 3/4 plus 1/4 inch middling from the drum and pulley type separators. Their product is 1/4 inch. The rolls are 40 by 15 inches.

Belt-type separator middling, minus 1/4 inch, goes to 40 by 15 inch rolls for further reduction. These rolls are run face to face, and the nominal product is 10-mesh, although much of it is finer.

Before the making of highest-grade concentrate started, these rolls were sufficient to handle the ore in process of separation, but since that time the necessity for fine grinding has required the introduction of more rolls into the flow sheet.

Taking No. 5 mill as an example, the last set of 40 by 15 inch rolls not only reground the minus 1/4 inch material coming from the belt-type separators in the "old mill" but also rehandled the middling selected by the belt machines in the annex - material which had already gone through these same rolls. At present, a program of providing rolls for the annex material alone is being pushed and half of them are in operation.

The rolls are located on the ground floor of each mill. They are arranged symmetrically along the longitudinal axis of the mill so that the rolls of the same class occupy the same relative position on each side of this axis. It is thus possible to run only half of the mill at a time.

The rolls do not run in closed circuit with screens. A conveyor (running the full length of the mill) is placed underneath the rolls on each side. Roll products are discharged directly onto this conveyor, which transports them to the elevator taking the dryer discharge. There the products join the dryer discharge and pass with it through the same flow sheet.

The simplicity of operation and the absence of many small units have caused the retention of this system instead of the adoption of the closed-circuit system of grinding.

Chrome alloy steel shells are used. When they have become grooved to such an extent that their operation is inefficient, they are removed and sent to the machine shop, where the faces are trued up. A shell which has thus been turned is placed where lighter duty is expected of it. The roll shells which have been working on 3/4 inch material are placed in the machine which treats minus 1/4 inch material and are there worn out.

During the past year, the large pulleys from some of the rolls have been removed and pulleys having the same diameter as the small wheels have been substituted. The fly-wheel effect of the large pulley is of course lost, but the saving in the bearings and the lessened tendency for the rolls to slip past each other and form a flange have more than offset this loss.

SHIPPING BIN

The shipping bin is a concrete structure 74 feet long by 13-1/2 feet wide, inside dimensions. It is flat bottomed with numerous draw-off points to reduce the dead contents. Mill concentrate, known as mill head, is delivered to the top of the bin by a 20 inch conveyor belt. It passes over four belt machines for the purpose of making a final cleaning. These machines make two grades of concentrate, No. 1 concentrate, which is the high-grade product, and No. 2 concentrate, which contains an appreciable amount of phosphorus. If there is no sale for the latter, it is returned to the mill to be mixed with the crude ore and goes with it through the regular flow sheet. This probably results in some of its being returned through the entire flow sheet several times, but the quantity is not sufficient to require a separate retreating plant.

The bin is divided into three compartments; the center section is for No. 2 concentrate, and the remaining two for No. 1 concentrate.

The bin spans a 100 ton Fairbanks railroad scale. This scale is periodically checked and adjusted. Its accuracy is maintained within the same limits as those of the weightometer on the crude-ore conveyor belt. The weight of concentrate as determined by this scale is used in obtaining the ratio of concentration and the recovery by the mill.

TAILING DISPOSAL

Tailing in a dry state is conveyed from the mill to a pile. During the summer months much of the tailing produced is sold as crushed rock for construction purposes. For the purpose of sizing the tailing into finished products, a screening plant has been inserted into the disposal system.

The first conveyor from the mill discharges into a trommel which divides the stream into minus 2 plus 1-1/4 inch, minus 1-1/4 plus 3/4 inch, minus 3/4 plus 1/4 inch and minus 1/4 inch sizes. These products are known as No. 1, No. 2, and No. 3 rock and sand, respectively.

Each size falls into a surge bin which automatically discharges onto the conveyor belt going up the pile. Under normal conditions, these bins are drawn before becoming full, but should the cycle be interrupted for any reason, the overflow feature prevents blocking the screen.

The various sizes are conveyed from the surge bins to the rock shipping bin separately. There they go through another screening operation which insures accurate sizing of the products. The rock sizes are made in a trommel while the sand goes over a Hum-mer vibrating unit to remove fine material. At present the sand shipping product consists of minus 1/4 inch plus 40 mesh material. Because of the demand for sand, a separate conveyor has been provided for this product in order to prevent any loss through overflowing of the surge bin.

Four lifts are necessary to transport the tailing to the discharge point. Three bins are provided in the circuit, each bin being located at the greatest economical conveying distance from the one preceding it. Each bin also acts as a surge point, having a capacity of about 15 tons, enough to hold at least a half hour run of tailing, should there be trouble with any of the belts following. The last belt is driven at the tail pulley.

The tailing shipping bin spans a spur of the railroad and can load cars on this track or load trucks and wagons from the side of the bin. Each compartment holds one railroad car.

This crushed stone and sand is ideal for concrete aggregate, having sharp edges and being inherently strong. There is no vegetable matter in the product. (The aggregate in the concrete of the piers and approaches of the Lake Champlain Bridge is composed of Mineville crushed stone).

Such tailing as is not needed for immediate shipment is loaded by size into railroad cars and taken to a stocking yard which supplements the shipments direct from the mill. The yard acts as a surge point and permits immediate shipment of rock when the customer's demands are in excess of mill capacity.

CONVEYORS

The conveyor equipment of No. 5 mill is as follows:

(1) The 20-inch belt conveyor taking crude ore from the hopper beneath the lorry car and discharging to two 5K crushers. This conveyor passes over a Merrick weightometer which automatically weighs the ore going into the mill. It is driven by an individual 25 hp. motor.

(2) One 20-inch belt conveyor carrying the undersize from the first screen to the top of the dryer.

(3) One 20-inch belt conveyor taking the dried ore from the discharge box of No. 2 elevator and delivering it to two trommels which are above the magnetic separators.

(4) Two 20-inch belt conveyors, one on each side of the mill. These conveyors run parallel to the long axis of the mill, beneath the coarse rolls. They collect the roll product and carry it to No. 2 elevator, where it joins the dryer discharge. (Product is middling).

(5) The conveyor (4) under the rolls on the east side of the mill discharges to a 20-inch cross belt conveyor which carries the material to No. 2 elevator.

(6) Two 20-inch belt conveyors. There are cross conveyors collecting the product from the fine rolls which treat middling from the belt machines, and discharging to No. 1 annex elevator.

(7) One 20-inch belt conveyor, on the conveyor floor, along the longitudinal axis of the mill, running underneath the magnetic separators to receive concentrate which is discharged into the concentrate elevator.

(8) One 20-inch belt conveyor receiving concentrates discharged from the concentrate elevator and carrying it across the railroad tracks to the shipping bin.

(9) Two 20-inch belt conveyors, one on each side of the mill, underneath the magnetic separators to receive tailing which is discharged to (10).

(10) One 20-inch belt conveyor carrying tailing from the mill to the sized-rock screening plant.

(11) Two 20-inch belt conveyors carrying sized rock and sand to the shipping bin.

(12) Four 20-inch belt conveyors, comprising the four lifts on the tailing pile, disposing of all the tailing or such portions as are not shipped.

ELEVATORS

A description of the elevators in use at No. 5 mill is given in the following paragraphs:

(1) One No. 1 elevator (inclined); 30-inch continuous bucket belt elevator. It consists of buckets 30 inches long by 15-1/4 inches high by 10-5/8 inches projection fastened to a 32-inch by 8-ply belt. The elevator receives the discharge of the two 5K gyratory crushers plus the by-passed undersize from the gravity screen ahead of the crushers and elevates them to the hopper above No. 1 trommel.

(2) One No. 2 elevator (inclined); 30-inch continuous bucket belt elevator. This elevator is a duplicate of No. 1 elevator in physical properties. It receives the product discharged from the dryer, together with the return middling load from two 40 by 15 inch rolls. These products are elevated to the top of the mill and discharged into a hopper over the conveyor to the trommels.

(3) One annex elevator No. 1; 16-inch centrifugal discharge belt elevator, consists of buckets 16 inches long by 6-3/4 inches high by 6-5/16 inches projection fastened to an 18-inch 8-ply belt. This elevator handles middling from the 5/16 and 1/4 inch belt-type separators ground to nominal 10 mesh in two 40 by 15 inch rolls. The material is elevated to the top of the annex, where it is transferred to a box over the annex belt-type separators by a chute.

(4) One annex elevator No. 2, east side; 26-inch centrifugal discharge belt elevator, consisting of buckets 26 inches long by 6-3/4 inches high by 6-5/16 inches projection fastened to a 30-inch 8-ply belt. This elevator is designed to handle the low-grade concentrate from the old mill plus the return middling from the annex separators. This elevator will be duplicated by another on the west side of the mill for the same service when the annex is completed.

The belt is joined with a butt joint covered by a patch consisting of a length of belt sufficient to be covered by six buckets, three on each side of the joint. The patch is fastened to the belt and the buckets are then bolted on through both patch and belt.

Head pulleys are lagged and the belts are run comparatively loose.

A small, portable, electric drill and an electric tool for running nuts on the bucket bolts facilitate repairs to the elevators.

This description of the elevators is taken from No. 5 mill, but it applies, with some modifications, to No. 4 mill.

PERSONNEL

The mill force consists of:

1	Mill foreman	
1	Shift boss	
1	Boss machineman	
3	Machinemen	
2	Screenmen	
1	Motorman	
1	Lorry carman	
1	Dryer man	
1	Roll tender	
1	Oiler, roll floor	
2	Oilers, general	
3	Conveyor tenders	(1 - Feed roll, crude ore (1 - Chip and steel picker (1 - Conveyor floor
3	Millmen	
2	Scalemen	
2	Men, screening and loading sized tailing	
2	Men, tailing disposal	
2	Millwrights	
1	Electrician	
30		

The machinemen are responsible for the care of the magnetic separators. One of the three tends the machines in the shipping bin; the other two are in the mill proper.

Because of the magnetic qualities of the ore, it is not possible to pick tramp iron from the crude ore by magnetic means. This necessitates the use of a chip picker on the crude-ore belt.

Mill men are classed as general labor. They are used principally for cleaning up. The scale men spot, load, and weigh out concentrate cars from the shipping bin.

During the summer months, when there is a large demand for sized tailing, this by-product requires two men for screening and loading.

SAFETY-FIRST WORK

At Mineville, safety-first work is largely delegated to the men. A general committee composed of the general superintendent and department heads has supervision over all safety-first work. This committee has no set meetings, being called together only on matters of great importance and to discuss new programs.

Working in conjunction with the general committee are the various subcommittees composed mainly of the men themselves. Six subcommittees are responsible for the conduct of safety work. These six committees are named for the working place of the men composing them, as follows: Old Bed mine, Harmony mine, New Bed mine and No. 4 mill, No. 5 mill, shops, and general surface.

The foreman of each unit is the chairman of his subcommittee. He and the secretary are permanent members; all others rotate. The men are selected by the foreman so as to represent the various phases of work under his jurisdiction. One third of the membership drops out and is replaced by a new group each month. By this plan, every man of the permanent force will at some time have served on a committee.

During his term of service, a member is responsible for safety conditions in the area he represents. This plan seems to give added interest to the work. Where the former attitude was to let the foreman and bosses look after safety work, the men now attend to much of the immediate detail themselves. They are encouraged to bring safety matters promptly to the attention of the foreman and in cases requiring immediate attention to take corrective measures at once. There has been no observable tendency to abuse authority.

Meetings of the subcommittees are held twice a month. Most of the suggestions discussed are disposed of at these meetings either by acceptance or rejection. Some suggestions are referred to the general committee. These include suggestions calling for new equipment or suggestions which properly should be handled by some other subcommittee. The head of the department is present at these meetings as an observer and contact member between the general and subcommittees.

At each meeting, in addition to safety suggestions, the accidents occurring between meetings and the means of preventing them are discussed.

Results have proved very satisfactory. There are no mass meetings. The safety idea has been spread through the men who have served on safety committees.

The first reflection on the safety work was a large increase in frequency and a decrease in severity of accidents. The trend is now a decrease in both frequency and severity.

HEALTH

In spite of the dusty condition of the mills, casualities from pulmonary diseases are very few. Loss of time traceable to colds is small. The explanation has been offered that there is a comparatively small amount of siliceous material in the crude ore and much of this is thrown out in the coarser sizes. The magnetite is not harsh nor penetrating. Men are encouraged to wear dust protectors, but are not required to do so.

SAMPLING

A sampler is stationed at each mill for the purpose of furnishing quick and reliable information to the foreman in charge. He is not part of the mill organization. The head chemist is responsible for the taking of samples and the routine followed.

As soon as the mill starts in the morning, samples are taken of the No. 1 concentrate, mill concentrate, or feed to the final cleaning machines and tailing. Results of these samples are available to the foreman, usually within a half hour.

Hourly samples of tailing are taken. Three samples are combined, sent to the laboratory, and the results returned to the foreman for his control of the mill. Mill concentrate or mill head is sampled every hour for mill control.

Each car of the shipping product is sampled at a point one-third the height of the pile in the car, measuring from the base of the pile. This point was selected as the result of many trials to secure truly representative samples. Fourteen places are reached on each car, six along each side and one at each end. Experience has shown that the height of the sample point is less important when the concentrate is finely ground than where there is a wide divergence in the size of magnetite particles. This is due to the natural segregation of sizes on the heap which is less pronounced where the finer sizes are concerned, since they more nearly approach uniformity in this respect.

Frequent sampling of the individual magnetic separators is followed to check up on the performance of each unit, although not as a daily routine. This practice is invariably used when the routine control samples show that iron losses in the tailing are high or that the iron content of the concentrate is low.

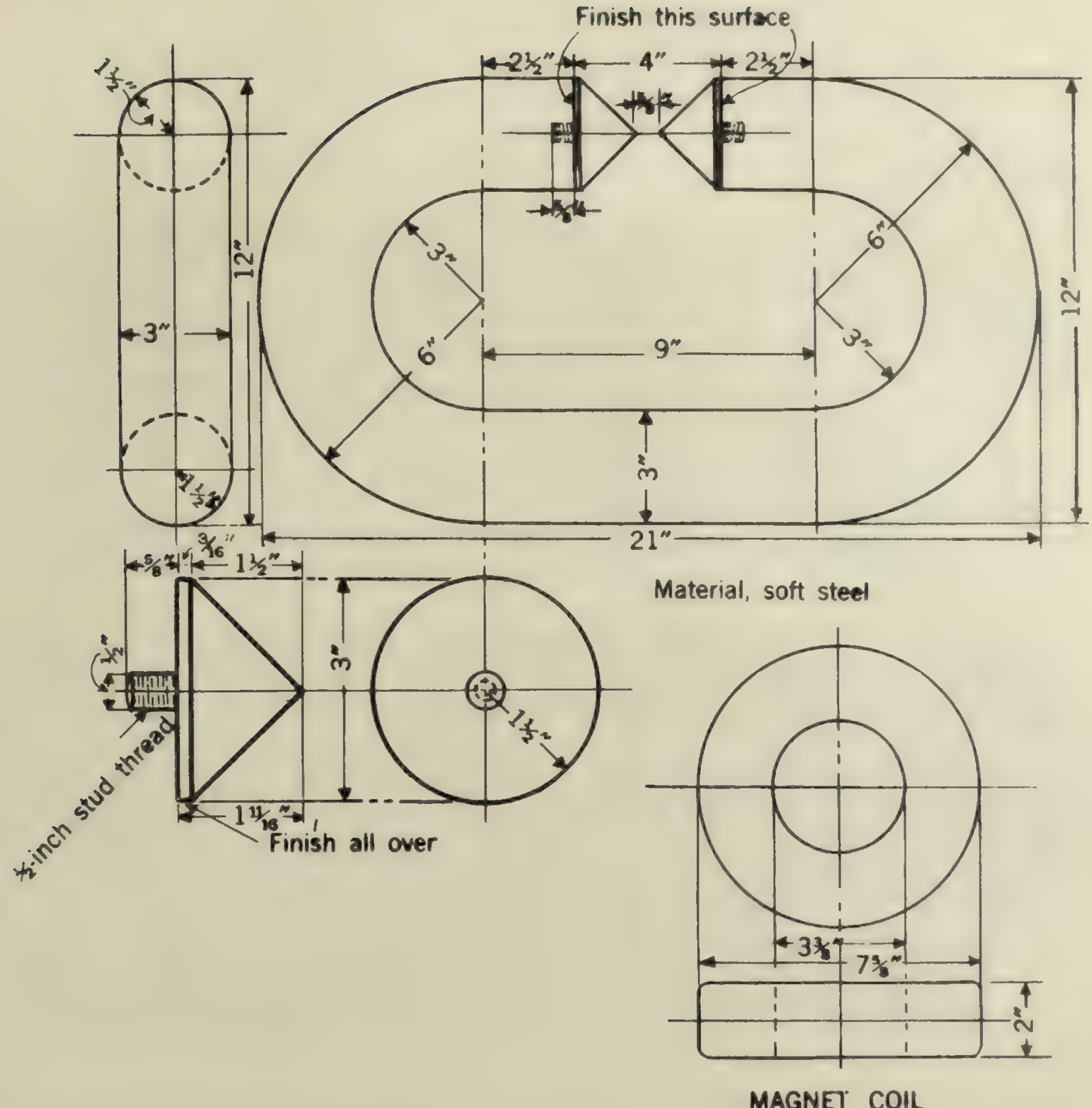

Figure 10.—Laboratory magnet

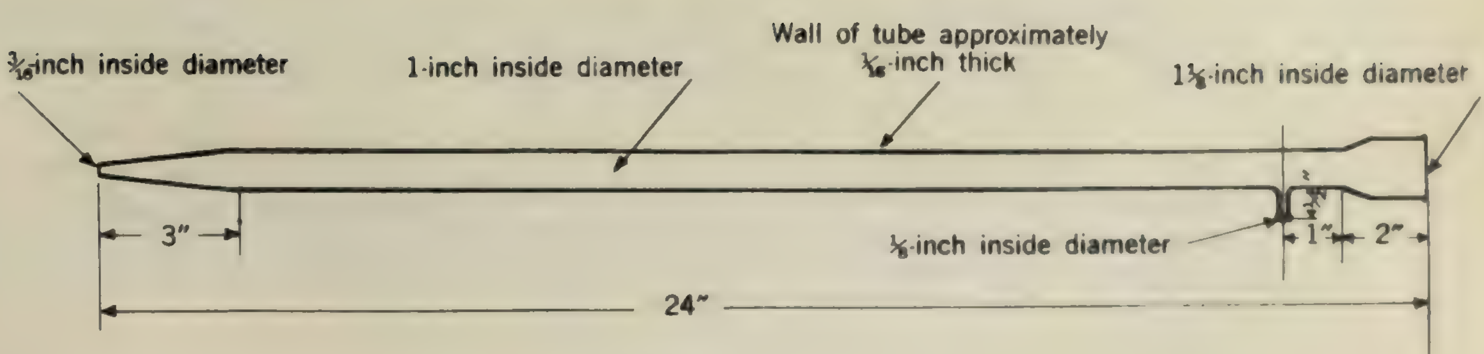

Figure 11.—Glass tube for magnetic iron determination

Magnetic iron determinations are made for mill control. The apparatus used consists essentially of a powerful electromagnet with pointed poles (fig. 10) and a long glass tube (fig. 11) having facilities for introducing the sample and wash water. Four coils are required in the construction of the magnet. Each coil consists of 1,500 turns of No. 17 double cotton covered wire, equivalent to 17 pounds of wire per coil, and is insulated by a wrapping of one layer of oiled silk and one layer of friction tape painted with insulating compound or shellac. The four coils are connected in series and operate at 110 volts and 225 amperes d. c.

The tube is placed between the poles of the magnet, and water flowing through it is agitated with short, vertical strokes. Magnetic material, of course, is held between the poles while gangue is washed out. When the wash water is clear, the sample is transferred to a beaker, excess water is decanted off, and the residue dried. The dried residue is weighed and the proportion of the weight of concentrate from the sample to that of the original multiplied by a constant gives the magnetic iron content of the sample. A 1-gram lot is taken for concentrate determinations and a 5-gram lot is taken for tailing determinations. This latter weight is chosen so as to decrease proportionate error in weighing the small quantity of iron in the concentrate from the sample.

Crude ore is not sampled as part of the routine. Its iron content is determined mathematically during the monthly recapitulation of concentrate and tailing results. The grade of crude is governed by the ratio of concentration.

Sampling of lump ore calls for experience and good judgment on the part of the sampler. Picking the sample at measured points on the car rarely gives representative results, and samples are usually taken at random within specified areas.

In cutting samples from the head pulley of a conveyor belt, the sampler pushes a pan directly into the stream. The same procedure is followed when sampling at magnetic separators.

Concentrate is sampled from the cars with a small trowel. The surface of the heap in the vicinity of the sample is brushed with the hand to remove the dust which accumulates and which has a tendency to salt the sample. The dust is a result of milling operations and is high in phosphorus.

Preparation of the sample consists of crushing it in a jaw crusher, reducing it still further in rolls (both these pieces of equipment being of laboratory size), riffling, and finally pulverizing. In the case of concentrate, the first two operations are not necessary, and the first step becomes that of riffling.

Table 2 presents the milling cost of a ton of concentrate at No. 4 and No. 5 mills combined.

Table 2. - Summary of Costs

Concentrators: No. 4 and No. 5. Year, 1930.

Method: Magnetic concentration.

Ore treated, 905,405 tons Concentrate produced, 538,253 tons

	Labor 1/	Supplies 2/	Total
General 3/....................	$0.054	$0.015	$0.069
Crushing and grinding.........	.019	.028	.047
Drying and separating.........	.030	.031	.061
Screening.....................	.015	.011	.026
Elevating and conveying.......	.023	.022	.045
Power and transmission 4/.....	.017	.053	.070
Loading from bins.............	.014	.006	.020
Transferring Harmony ore 5/...	.009	.047	.056
	0.181	0.213	0.394

Note: The ratio of concentration at No. 5 mill for 1930 was 1.396 to 1 and at No. 4 mill was 1.913 to 1.

(1) Labor charge includes direct operating and repairs cost.
(2) Supplies charge includes trucking.
(3) Includes superintendence, heating, safety, and other items not properly chargeable to one account.
(4) Power was not distributed to the various accounts.
(5) Includes freight charges by the railroad for transferring ore from Harmony to New Bed and rehandling charges in the New Bed mine for this transfer, necessary because No. 3 mill is out of commission.

NO. 3 MILL

The mill being planned to replace No. 3 mill will have a flow sheet (fig. 12) similar in general to that employed at the other mills, except that provision is made for up-to-date machinery and the elimination of complete drying of the ore in the coarse stages of separation.

The mill will be erected on the site of the building which was destroyed by fire in 1925 and in so far as possible will use the old building. No. 3 mill is located at the Harmony A shaft headframe and will treat crude ore from that mine.

Skips from the mine will dump into a bin feeding a No. 8K Gates gyratory crusher, the 4-inch discharge product falling on a 36-inch belt conveyor. A Merrick weightometer will automatically weigh the crude ore going into the mill.

This conveyor will transport the ore to a 600-ton draw-off capacity surge and storage bin from the bottom of which the ore will be fed by a shaking-plate feeder to a continuous bucket belt elevator carrying 36-inch buckets.

The elevator will carry the ore to the top of the mill discharging into the feed box for two 48 by 12 foot trommels which will size it into minus 1-inch; plus 1-inch; minus 2-1/2 inch; plus 2-1/2 inch; minus 4-inch.

Plus 2-1/2 inch or oversize material will go to pulley-type separators which will effect a tailing-middling separation. The tailing will go to waste or sized stone storage. The middling will fall through chutes to two 6K Gates gyratory crushers. The crushed product will fall onto the shaking plate feeder from the bin and return to the top of the mill with the fresh crude ore.

Minus 2-1/2 inch material will fall to pulley-type separators to make a tailing-middling separation. Middling will be conveyed to a 7 foot Symons cone crusher set a 1/4 inch opening for further reduction. The crusher product will be elevated to the top of the mill in a 36-inch centrifugal discharge elevator, and discharged by conveyor into bins over a battery of eight double-drum magnetic separators.

Minus 1-inch material will be conveyed directly, without preliminary separation, to the top of the dryer which was used in the old mill. This dryer is of the conventional tower type used in this district. The dried product will be elevated by a 36-inch continuous bucket belt elevator to the top of the mill, to two 8-foot, type 39 Hum-mer screens to make a plus and minus 1/4-inch sizing.

The minus 1/4-inch screen product will be conveyed to bins above a second battery of eight double-drum separators. Plus 1/4 minus 1 inch material will go to double-drum separators for a tailing-middling separation. The middling product will join the middling from the pulley-type separators working on minus 2-1/2-inch material and go to the cone crusher.

The separators will be the new double-drum type, making concentrate, middling, and tailing. Tailing will go to waste. Middling from both batteries of separators in this section of the mill will go to a set of 72 by 24 inch rolls for reduction in size. The roll product will be elevated to the top of the mill in a 36-inch centrifugal discharge elevator to join the minus 1/4-inch material from the Hum-mer screens. Provision will be made for dividing the discharge from both roll and cone crusher elevators so that it may go to either battery of separators or both, assuring an evenly distributed feed to all machines at all times. The middling from these machines will not leave this section of the mill until separated as concentrate or tailing.

At this point, another change from the old flow sheets takes place. As stated elsewhere, high capacity with clean tailing may be obtained when making a low-iron concentrate from a medium-grade crude; or high capacity with high iron concentrate and a clean tailing will be obtained with a rich feed to the machine.

The concentrate from the 16 separators of the two batteries working on crude ore will average approximately 60 per cent of Fe. It will go by elevator to a battery of 10 double-drum magnetic separators in what will be known as the annex, for the purpose of raising the iron content to 66 per cent or better. This concentrate will go to the shipping bin where it will be passed over belt machines for final cleaning. The middling from these machines will return to two 72 by 24 inch rolls set to run face to face for further reduction in size, and will ramain in the annex circuit until separated, either as concentrate or tailing.

Provision has been made for the erection of two dryers to dry the mill concentrate before separation on the annex machines should experience prove they are not dry enough to give the desired results.

The shipping bin will contain four belt-type separators for making a final cleaning of the concentrate. With these machines it will be possible to produce a highgrade concentrate with a comparatively lean middling, which will be returned to the mill for retreatment, or to make two grades of shipping product, as occasion demands.

It will be noticed that only two sets of screens are used, the two trommels and two 8-foot Hum-mers. These screens are used in the preliminary stages, before any concentrate is made. Proper regulation of current impressed on the magnets will accomplish the necessary sizing.

It is believed that the dusty condition of the mill will be kept to a minimum by not completely drying the ore in its initial stages. Should it be necessary to redry it before treatment in the annex, the resulting dust will be confined to a comparatively small area and can be handled effectively by a dust-collecting system.

With the installation of larger and more modern equipment in this mill, it is expected that the Harmony crude will be treated at a much lower cost.

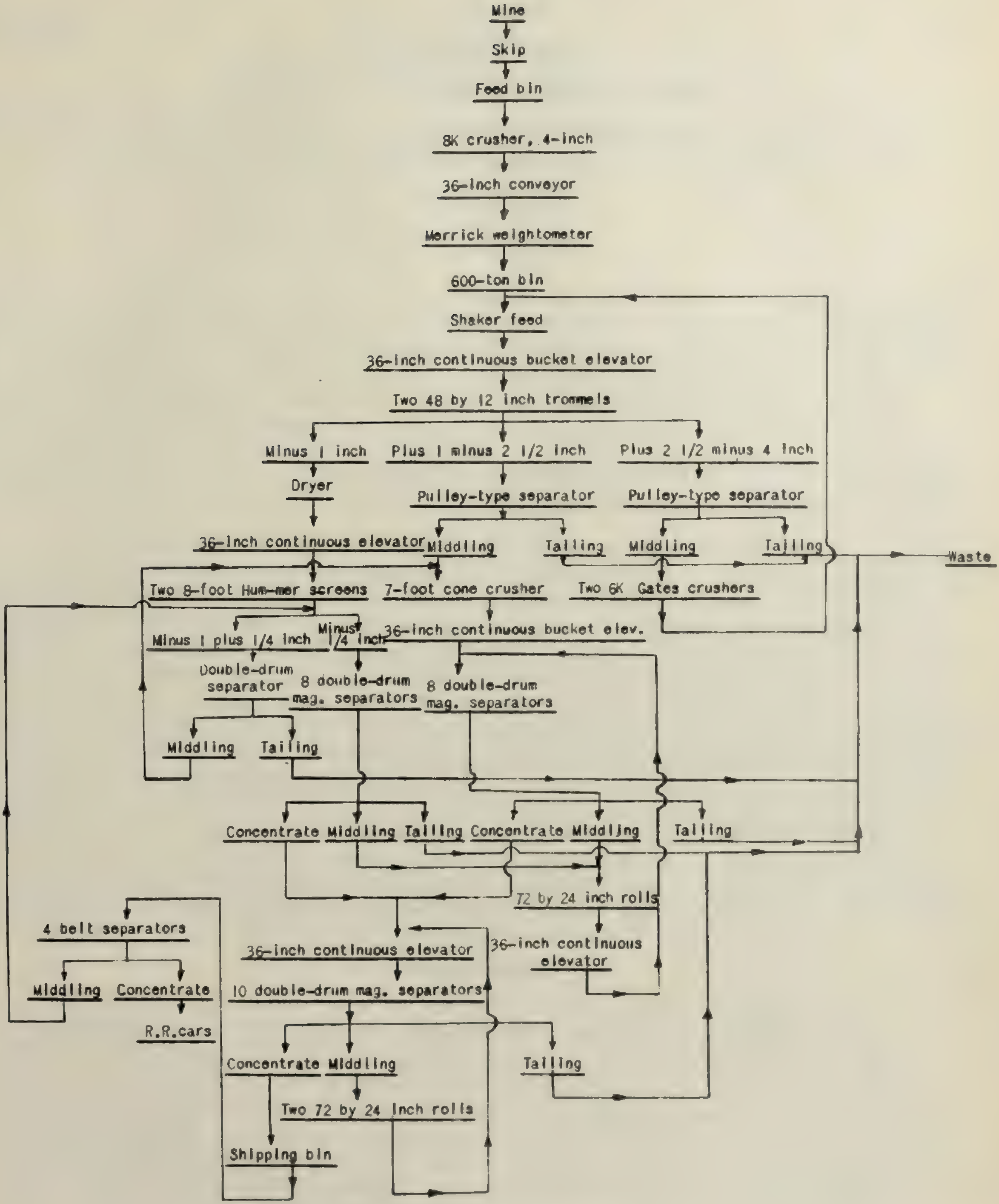

Figure 12.- Flow sheet of mill No. 3

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

BONUSES TO ENCOURAGE SAFE WORK AND FOR WORK SAFELY DONE[1]

By D. Harrington[2]

PURPOSE OF THIS PAPER

There has been much writing and argument on the subject of giving bonuses to miners, quarrymen, metallurgical and chemical workers, and others as a stimulus for safe work and for work safely done. Theoretically, everyone should be careful and should not need monetary or any other type of encouragement for the prevention of accidents; as a fact, few people are careful, and this is primarily the reason why many if not most accidents occur. A bonus, however, is not simply money given away by an employer; it must be earned, so to speak, before it becomes due; in other words, if the standard established is not reached, no bonus is paid, and in many cases some disciplinary action is taken.

The United States Bureau of Mines is asked from time to time for suggestions for rewards to mine workers who have a good attitude toward working safely or who need some stimulus to cause them to make greater or more earnest efforts toward safety for themselves and their co-workers. A recent inquiry prompted the collection of data on some safety bonus systems from reports of this bureau's field engineers and others, and some of these are now given. Bonus practice at both coal and metal mines is included because a system used at metal mines is generally adaptable to coal mines; in fact, in numerous cases, bonus systems recently started at some coal mines are based on systems in vogue and giving satisfaction at some metal mines; in some instances, also, metal mines have been able to get good suggestions from coal mines or from the cement or petroleum industries. It should be understood that a bonus, even as applied to the encouragement of safety, does not necessarily mean payment of money; in fact, some successful bonus systems have in them no direct money exchange; hence, the bonus as treated in this paper is that defined as "something given in addition to what is ordinarily received by, or strictly due to, the recipient."

ACKNOWLEDGMENTS

Acknowledgment is given here to members of the safety division of the Bureau of Mines who sent in some of the information here presented; to the Engineering and Mining Journal and Mining Congress Journal, from which some items were abstracted; to various mining officials who gave data in one form or another; and to M. von Bernewitz, of the United States Bureau of Mines who aided in framing parts of this paper.

1 - The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6625."

2 - Chief Engineer, Safety Division, U.S. Bureau of Mines, Washington, D.C.

COAL MINES

Alabama

At the Alabama Fuel and Iron Co's coal mines in Jefferson and St. Claire Counties, fire bosses check the men into and out of the mines and search them for matches, pipes, cigarettes, and other smokers' articles every day. The fire bosses are paid $5 for every match found in a workman's pocket at the entrance to the mine. However, $5 is deducted from the fire boss's pay for every match found inside the mine.

This matter of trying to prevent open flames in closed-light mines is one very difficult to handle, and if this or any other bonus system is effective in preventing smoking in mines in which smoking is prohibited by rule, regulation, or law, the bonus system will have done a remarkably good deed.

Illinois

1. The Bell and Zoller Coal Co. operates mines at Ziegler, Centralia, and Peoria, Ill., and among numerous safety practices pays a bonus to foremen. On January 16, 1932, nearly $2000 was distributed as prizes to officials of the mines for safe work during 1931, as follows:

Bonuses paid to officials of the Bell and Zoller Coal Co. mines for safe work, 1931

Grand individual prize for hand-loading mines

Official and prize	Record
Top boss, $150	5,347 man-shifts; no lost time
Top boss, 75	16,320 man-shifts; 2 accidents, 11 days lost
Assistant mine manager, $50	4,658 man-shifts; 1 accident, 14 days lost
Face boss, 25	14,474 man-shifts; 11 accidents, 134 days lost

Grand individual prize for mechanical loading mines

Loader boss, $115	3,030 man-shifts; no lost time
Chief electrician, 92.40	2,435 man-shifts; no lost time
Loader boss 67.60	1,781 man-shifts; no lost time
Loader boss 25.00	1,970 man-shifts; 1 accident, 1 day lost

Bonuses paid to officials of the Bell and Zoller Coal Co. mines for safe work, 1931 - Continued

Mine prizes for frequency rate based on lost-time accidents (3 mines)

Official and prize	Record
10 bosses, $15 each	- - -
11 bosses, 10 each	- - -
28 bosses, 5 each	- - -

Mine prizes for frequency rate no lost-time accidents (3 mines)

10 bosses, $15 each	- - -
4 bosses, 10 each	- - -
11 bosses, 5 each	- - -

Mine prizes for severity rate no lost-time accidents (3 mines)

28 bosses, $15	- - -
9 bosses, 10	- - -
11 bosses, 5	- - -

Many others deserve honorable mention because a narrow margin determined some winners.

For the safety contest for 1932, the prizes will be as follows:

Individual Prizes

a. Quarterly prize of 1 cent per man-shift to bosses in groups 1, 2, and 3 who go through quarter of the year without a lost-time accident.

b. Quarterly prizes of ½ cent per man-shift to bosses in group 4 who go through a quarter of the year without a lost-time accident.

c. A yearly prize of $20 to every boss who goes through the year without a lost time accident.

d. A yearly prize of $20 to the boss in each group who has the best rating at the end of the year in frequency rate no lost-time accidents.

e. A yearly prize of $20 to the boss in each group who has the best rating at the end of the year in frequency rate lost-time accidents.

f. A yearly prize of $20 to the boss in each group who has the best rating at the end of the year in severity rate lost-time accidents.

Mine Prizes

a. A yearly prize of $10 to each boss employed at the mine which has the best rating at the end of the year in frequency rate no lost-time accidents.

b. A yearly prize of $5 to each boss employed at the mine which is second in the rating at the end of the year in frequency rate no lost-time accidents.

c. A yearly prize of $10 to each boss employed at the mine which has the best rating at the end of the year in frequency rate lost-time accidents.

d. A yearly prize of $5 to each boss employed at the mine which is second in the rating at the end of the year in frequency rate lost-time accidents.

e. A yearly prize of $10 to each boss employed at the time which has the best rating at the end of the year in severity rate lost-time accidents.

f. A yearly prize of $5 to each boss employed at the mine which is second in the rating at the end of the year in severity rate lost-time accidents.

2. The Union Colliery Co. at Dowell, Ill., is paying a bonus to its employees and officials. The company checked past accident records for a number of years and determined the average cost per ton, and any reduction in this cost per ton is divided between the company and the mine officials and employees; the company takes half and divides the other half among the mine officials and employees. Five periods of three months each have brought the following results:

Bonus paid for reduction in average cost per ton of production, Union Colliery Co.

Period	Foremen - number and amount paid each	Men-number amd amount paid each	Total savings	Employees' share of savings
Oct. 1-Dec. 31, 1930	4 and $100	127 and $22+	$6,589	$3,396
Jan. 1-Mar. 31, 1931	5 and $106	124+ and $24	7,048	3,524
Apr. 1-June 30, 1931	- and $25	- and $5	3,200 (loss)	Although not earned earned because of 3 fatalities, $785 was paid.
July 1-Sept.30, 1931	-	-	2,681	$1,341
Oct. 1-Dec. 31, 1931	-	-	-	$415 only, because of a fatality.

The officials of the company knew that accidents were costing about 5 cents per ton, and it was figured that if this were reduced to 2 cents or less, the company could well afford to share the amount saved. At the end of the first 12 months trial it was decided that the general results had been good, but there was yet much room for improvement. The savings totaled $13,119 and the men's share was $8,944, though they were paid more than this, but the net result was a definite reduction in accident occurrence coincident with a dollars and cents gain to both employer and employee.

The last three months of 1931 worked out as follows, the amount saved being reduced considerably because of occurrence of one fatality:

Total coal, tons	201,278
Standard compensation cost and expense, at 5 cents	$10,064
Actual compensation and accident expense estimated	$ 9,233
Total saving compared with standard cost	$ 831
Men's share, 50 per cent	$ 415

Pennsylvania

An excellent form of bonus is that of the Hudson Coal Co. of Pennsylvania, an anthracite producer which, as the reward for having an outstanding safety record during the first nine months of 1931 sent seven officials to the annual convention of the National Safety Congress at Chicago, October 12 - 17, 1931.

The W. J. Rainey Co., operating bituminous mines in western Pennsylvania, has also sent delegations of mine officials to meetings of the National Safety Council as a reward or prize or bonus for good safety achievements. As may be seen, this form of bonus should amply repay any mining company, coal or metal. The fortunate officials are able to participate in the numerous safety discussions, to get copies of the papers presented, and learn new safety phases from them as well as from the discussions; moreover, they can mingle with mining men from all parts of the country, and coal mining men can get good suggestions from metal mining men and metal mining men can profit from the coal men. Moreover, they can see the practical exhibits, and this in itself amply justifies the time and expense of the trip. A trip to the next National Safety Congress as a reward for some definite safety performance such as a 50 per cent reduction in accident frequency or severity, or the operation of a mine for a month or several months without a lost-time accident, would assuredly be profitable not only in the avoidance of the numerous human miseries entailed when accidents occur, but also in the saving of definite amounts of dollars and cents which are almost certain to accrue when those employed in and around mines are given specific rewards for the avoidance of accidents.

Utah

The United States Fuel Co. employs a bonus system which apparently works out satisfactorily. The mine foreman receives a $10 bonus for a 3-month period for the absence of a single lost-time accident during that period. The operation is divided into departments, and the sub-foremen in charge of each department receive a $5 bonus for a 3-month period without a lost-time accident. For the second consecutive period without a lost-time accident the bonus is doubled. After the second consecutive 3-month period without an accident the bonus remains at the same amount as for the second consecutive 3-month period. In other words, the bonus is not increased beyond the second period of three months, regardless of how long the mine or department may operate without a lost-time accident. That this company is reaping good results from its bonus and other safety measures may be inferred from the fact that its Panther mine has not had a fatality since 1925, though for much of this time production has been largely (and in 1931 was wholly) from pillar extraction; its King 1 mine working in high coal has not had a fatality in nearly two years, and its King 2 mine has not had a fatality in nearly 2½ years. This company operated its three mines (with about 500 employees) in 1931 without a fatality and with but 27 lost-time accidents in the production of over 500,000 tons of coal, largely from pillar operations in high coal under heavy cover.

West Virginia

At its Dehue mine in West Virginia, employing about 375 men, the Youngstown Sheet and Tube Co. does a considerable number of things to bring about safe operation. The following method of rewarding men who work safely is rather unique and it has definite psychological value. The mine, which produces 2500 tons a day, is divided into sections, and if a section works a month without a lost-time accident, all men in that section are given two theater tickets. What actually happens is, the miners' children generally get the tickets, and if they do not get them each month they want to know who was injured and why!

That this mine is "on the right track" in its various activities with intent to bring about safety may be inferred from the fact that this mine worked from January 8, 1931, to and through January 8, 1932, without a fatality or any kind of lost-time accident and in this period produced 261,924 tons of coal with 378,874 man-hours of exposure to its approximately 375 employees.

Wyoming

In an effort to improve its accident record, the Union Pacific Coal Co., which operates one mine at Hanna and seven mines in the Rock Springs district, Wyoming, employs approximately 2,000 men, and can produce about 11,000 tons of coal a day, has divided its mines into districts. Each district is in the hands of a foreman who is charged with the workers' safety, and each district is charged with each lost-time accident and credited with man-shifts worked. The number of man-shifts per lost-time accident was computed at the end of six months for 1931 (thereafter this contest will be for the calendar year), and the right to draw for two automobiles is given to every person who works in the districts which go through the given period without a lost-time accident. When the owners of the machines are

finally determined by drawing, a suitable legend is painted on the sides of the cars to certify that they have been awarded for safety, thus making the two cars continuous rolling advertisements for the cause. The foremen of the two sections represented by the winners of the cars are awarded $150 and $100 respectively. During the second half of 1931, nine sections with 511 men worked without a lost-time accident, and the awards were given at Rock Springs, Wyo., on February 26, 1932, when about 1,000 persons were present. The first prize was a Dodge sedan and was drawn by an employee of No.4 mine, Rock Springs, the foreman of the section getting $150 in gold; the second prize, a Chevrolet sedan, was drawn by an employee of E mine, Superior, Wyo., the section foreman getting $100 in gold.

This bonus causes the bosses, as well as each individual worker to be keenly interested in the avoidance of lost-time accidents in his particular section. Although the system has been in effect for less than a year, it is said to be eminently successful, not only in the avoidance of the misery of various kinds which comes from accidents but also as to the financial aspects, which are encouraging to all who are intimately concerned. This company now appears to be reaping good results from its accident prevention activities; its No.4 mine, Rock Springs, Wyo., has not had a fatality from April 16, 1923, to Dec. 31, 1931, with production of 2,476,122 tons of coal by an average of 216 employees; and its C mine, Superior, Wyo., has not had a fatality from July 5, 1927, to Dec. 31, 1931, with production of 1,019,755 tons of coal by an average of 133 employees.

Metal Mines

In "The Safety Bonus in Metal Mining," United States Bureau of Mines Report of Investigations 2617, 1924, F. C. Gregory cites the following four examples of bonuses for shift bosses:

> 1. Each shift boss who supervises 2,500 shifts without a lost-time accident shall receive a bonus of $30. No penalty for an accident shall be imposed except that all credit up to the day of the accident is lost; the new bonus period starting on the following day. A yearly bonus of $100 shall be given to the foreman having the best record for the year. Seriousness of accidents, if disability is beyond day of injury, is not considered.
>
> 2. A monthly bonus shall be paid to both foremen and shift bosses for accident prevention, with 1,000 shifts as a base. For shifts above or below this number, the bonus is in proportion.
>
> For foremen supervising 1,000 shifts:
>
> No bonus when lost time exceeds 1½ per cent of time worked.
> With no lost-time accidents $50
> With lost time from accidents less than ½ per cent of time worked 40
> With lost time from accidents less than 1 per cent of time worked 30
> With lost time from accidents less than 1½ per cent of time worked 25

Shift bosses shall receive one-half this bonus for the same records.

3. A bonus of $25, to each shift boss working 1,000 man-shifts without a serious accident and no lost-time accidents, graduated down to $7.50 and for no serious and not more than three lost-time accidents. After a serious accident to one of his men, the shift boss must work 500 man-shifts before starting on bonus period again. For preventing the report of an accident the shift boss shall be penalized 1,000 shifts for the first offense and debarred from bonus for a second offense.

4. A bonus of $10 shall be paid to each shift boss working 750 shifts without a serious accident; $5 per month shall be paid each jigger boss in charge of ten or more men who works the month without a serious accident.

These four plans represent the extremes of the conditions that are required before the bonus is earned. Under each plan the bonus is earned at frequent intervals, which is one of the requisites of its success. If the conditions are so severe that they can not be met by a sincere effort on the part of the bosses, the bonus will not have the desired effect. The safety bonus has resulted in a reduction in the cost of inspection by the safety department.

Arizona

1. Accident prevention is an important feature of Phelps Dodge operations in Arizona, New Mexico, and Mexico. Men who work for specified periods without injury are given engraved safety buttons of different design for one, two, and three years; for five years they receive an engraved belt and buckle. Foreman and shift bosses who run a month without injury to their men receive a bonus.

This company has had remarkable success in its efforts to reduce accidents, and while the bonus system is but a part of the rather extensive scheme of accident prevention, its effect is unquestionably good. In its metal mining branches (Old Dominion Copper Co., Globe, Ariz.; and Moctezuma Copper Co., Nacozari, Old Mexico) this company has reduced its accident rate per 1,000 shifts worked, as follows:

Year	Accident rate per 1,000 shifts worked.
1924	1.047
1925	0.629
1926	0.228
1927	0.122
1928	0.064
1929	0.038
1930	0.040

The accident rate per 1,000 shifts worked in 1930 was but about [illegible] per cent of the similar rate in 1924, or, in other words, this company has reduced its accident rate nearly 96 per cent in the 7-year period, 1924-1930, inclusive. Some of the details of the safety work of one of the branches of this company (the Old Dominion Co.) follow.

When in 1914 the Old Dominion Co., a well-known copper producer at Globe, now a subsidiary of Phelps Dodge Corporation, started its safety campaign, the shift bosses were told that improved conditions meant better work, that an indifferent companion should be shown what not to do, and that brain should be used as well as muscle. A system of bonuses, based on their safety record, was inaugurated. Each shift boss who worked 500 shifts or more a month without any lost-time accidents was given a bonus of $10. If there was any lost time and it was less than 1 per cent of the total shifts worked, a bonus of $5 was given. This proved to be successful as it encouraged the bosses to watch their men more closely. Every shift boss was entitled to this bonus each month, regardless of the number of accidents in the previous month. In this way all started a new month with a clean sheet. There was also a form compiled each month with their names, number of accidents, lost time, and other items. This made the bosses more careful because they did not care to see their names with many accidents opposite them.

The Old Dominion safety department was reorganized in 1925, and a system is now in force similar to that of the Phelps Dodge Corporation. The mine supervisory force, including general foremen, division foremen, and shift bosses, receive the following bonuses:

General foreman - - - - - -	$55 for clean sheet each month.
Division foreman - - - - -	$50 for clean sheet each month.
Shift bosses - - - - - -	$25 on basis of 1,000 shifts. The bonus can be more or less than this sum.

There is also a sliding scale of bonuses in effect, based on the percentage of time lost to total shifts worked, as follows:

Record	Foreman	Shift bosses
For clear record - - - - - - - -	$50	$25
Not more than $\frac{1}{2}$ per cent - - -	40	20
$\frac{1}{2}$ to 1 per cent - - - - - - - -	30	15
1 to $1\frac{1}{2}$ per cent - - - - - - - -	20	10
$1\frac{1}{2}$ per cent or more - - - - -	0	0

All records start anew each month.

As to other employes of the Old Dominion Co., when anyone has completed

six months service without a time-lost accident he is entitled to a safety button. At the end of two years, if this record is continued, he receives a different button, and at the end of five years he gets a gold and enameled button. With the button is issued a certificate showing how long he worked without a time-lost accident. Any man who leaves the company is also entitled to a certificate showing how long he worked without a time-lost accident. The periods are as follows: 150 days worked equals six months, 300 days one year, 600 days two years, and 1,500 days five years. Time off is not counted. If a man causes a time-lost accident to a fellow employee, this is counted against him, even though he himself is not injured.

With this company, it pays in more than one way to become interested and proficient in safety, because the crew of 15 men trained in mine-rescue and recovery operations who receive additional training every month and hold themselves for service available on short notice in case of fire or similar disaster receive a bonus of 30 cents a day, which may amount to about $7.50 a month.

Workmen who serve on the various safety committees of the Old Dominion Co. wear a safety button during the first three months and a safety watch fob at the conclusion of their term of six months. These buttons and fobs are presented by the company. Men take pride in serving on the committees and the system serves as a valuable means of interesting the men, also it creates many safety inspectors throughout the plant.

Michigan

1. The Cleveland-Cliffs Iron Co. has several bonus systems; shift bosses in whose section no lost-time accidents occur are paid 35 cents extra per shift. This urges them to greater efforts. All bosses are rotated to another section every three months so that one man will not always have a better region than another. Under this system of rotation certain bosses who had been regarded as better than others were found to be no better than the average, and a boss formerly regarded as ordinary or even below that grading might be found to be thoroughly competent when moved to other parts of the mine. Also the shifting around made the men more observant and self-reliant; rivalry was started; a boss shifted to a new section often made advisable changes which had escaped attention of his predecessor or predecessors.

For the mine workers in general, six months work without a lost-time accident brings each man a knife, and a year's safe work brings a 14-carat gold lapel button. Banner flags are awarded to a mine having an outstanding record and in addition the company has staged a monster picnic as a reward or possibly as a celebration for having had excellent safety records at its mining operations.

As a result of its numerous safety activities, of which the bonus is an important part, the Cleveland Cliffs Iron Co. has achieved a standing in safety in mining second to no mining organization in the United States. As of Dec. 31, 1931, this company had days of no lost-time operation at its properties as follows:

Cliffs A shaft, 176 days (this mine also has a record of 451 days of operation without a lost-time accident with 178 men employed for 516,648 man-hours in producing 235,976 tons of rock and ore).

Cliffs B shaft, 219 days (this mine also has a record of 466 days of operation without a lost-time accident, with 128 employed 511,184 ~~for~~ man-hours in producing 290,894 tons of rock and ore). (Both of the foregoing are underground mines).

Holman Cliffs, 643 days, in which 121 men in 640,120 man-hours produced 971,271 tons of rock and ore.

Hill Trumbull, 486 days, in which 85 men in 310,780 man-hours produced 318,240 tons of rock or ore. (The foregoing two properties are open pits).

Gardner-Mackinaw, 591 days, in which 103 men in 365,128 man-hours produced 157,004 tons of rock and ore. (This mine, which is an underground operation, also has a previous record of operation without a lost-time accident for 545 days with 102 men who in 393,338 man-hours produced 184,815 tons of rock or ore).

The Tilden mine, 784 days, in which 25 men working 176,265 man-hours in an open pit produced 424,053 tons of rock or ore and in addition moved 10,210 cubic yards of rock.

Athens mine (underground operation), 150 days; Spies Virgil (underground operation), 103 days; and Canisteo Cliffs (open pit), 238 days. This company now has six mines (three underground and three open pit), each of which has operated over one year without a lost-time accident.

2. The Copper Range Co., Painesdale, Mich., keeps a record of each shift boss and the subordinate trammer boss to show the number of man-shifts their men have worked without accidents. When a boss and his men have completed a year without accidents on the surface they receive a pocket knife, and if they work six months without an accident underground the boss and his men each receive a pocket knife. If the number of shifts worked on the surface is 9,000 before a calendar year is completed, the men are rewarded with the knives just as if they had worked a year, and when 4,500 shifts are worked underground they also receive the knives.

A second award is made to a boss and his men for completing 9,000 shifts underground or 18,000 shifts on the surface without accidents. This second award is a small gold lapel button with the words, "Copper Range Company, Second Award."

These awards have greatly stimulated the interest of bosses and men in safety and some excellent records have been established; the Champion No.1 shaft of this company operated from January 2, 1931, to January 2, 1932, without a lost-time accident and with an average of about 100 men employed for a total of 24,947 man-shifts;

John Hall, one of the underground bosses at Champion No.3 and No.4 shafts did not have a lost-time accident to the men working under him in 1931 during a total of 15,869-1/8 man-shifts; Philip Vincent, underground boss at Champion No.1 shaft did not have a lost-time accident to any of his men in 1931 during which 13,242 man-shifts were worked; and Cesar Campo, underground boss in the Baltic mine, did not have a lost-time accident to any of his men in 1931 during the 12,046½ man-shifts that they worked.

3. The M.A. Hanna Co., which in ordinary times operates 16 mines on the Menominee iron range, Mich., has an actively functioning safety system; it has community safety rallies followed by lunch, occasional safety dinners are given for the foremen, and prizes or rewards such as pocket knives, turkeys, and match holders have been given to encourage safety in operation. This company has numerous other safety devices and methods, so that the bonus can be given credit for only a part of the excellent results being obtained. In 1931 the company operated 10 properties and did not have a fatal accident, this being the first year in which no fatalities were suffered. In addition to having no fatal accidents in 1931, there were but 19 compensable accidents (loss of seven or more days) and but 27 lost-time accidents.

Some of the mines of this company have established remarkable safety records; its Wakefield open-pit iron mine at Wakefield, Mich., worked from December 1, 1927, to January 1, 1932, (a period of 1491 days), without a lost-time accident and with employment of about 113 men for a total of 1,288,869 man-hours. The Harold (underground) iron mine, Hibbing, Minn., operated with an average of 111 men for 964,634 man-hours from June 1, 1928 to December 31, 1931, or three years and seven months with but two lost-time accidents, totalling 95 days of lost-time. On December 31, 1931, this mine had operated without a lost-time accident for 696 days. The La Rue open-pit mine at Nashwauk, Minn., has a no-lost-time record of 442 days with about 126 men employed; the Mesabi Chief (open pit) mine has a no-lost-time record of 804 days with about 70 employed; the Susquehanna (open pit) mine, Hibbing, Minn., worked through 1931 with but one lost-time accident; the Rogers (underground) mine in the Iron River, Mich., region on January 1, 1932, operated 799 days with but one lost-time accident (44 days lost) and with 146 men working 643,143 man-hours; the Richmond (open pit) mine, Palmer, Mich., worked from January 1, 1928, to January 1, 1932, or four years, without a lost-time accident with 35 men employed for a total of 209,393 man-hours; the Hiawatha (underground) mine in the Iron River, Mich., region worked from August 7, 1930, to January 1, 1932, (or 511 days), without a single lost-time accident with 118 men employed for 306,486 man-hours; the Wabigon (open pit) mine, Buhl, Minn., operated from January 19, 1929, to Nov. 30, 1930, a period of 681 days, with but one lost-time accident (35 days lost) ~~per~~ 220,885 man-hours; and the Moroco (open pit) mine Thromalo Village Minn., worked from October 24, 1928, to November 30, 1930, or 767 days with but two lost-time accidents (28 days lost) to 56 men working a total of 403,151 man-hours.

4. If the Berkshire iron mine of Oglebay-Norton & Co. works for one month without a lost-time accident, each man is given candy, tobacco, or similar "delights." If the mine works for a year without a lost-time accident, every employee receives a pocket knife with his name on it. In addition, the Oglebay-Norton Co. has a safety trophy which is awarded to the mine that works three months without a

accident, and if two mines work three months without a lost-time accident, the trophy is awarded to the mine having the greatest number of man-hours; the awarding of a flag for safety accomplishment in competition among the various units of a company or even of a particular property is a type of "bonus" which is very likely to produce good results in the prevention of accidents, provided that there are definite rules governing all who are in the competition and provided that the rules are kept rigidly in effect. Not only in the Berkshire mine, but also at the Oglebay-Norton & Co. properties as a whole, has safety been kept to the front. The Berkshire mine won the Sentinels of Safety Trophy for underground metal mines in 1928 and 1929; in 1930 in the Sentinels of Safety Competition this mine ranked second, as it had no lost-time accidents with 283,807 man-hours of exposure, while the winner had 318,402 man-hours of exposure without a lost-time accident. The Berkshire mine was awarded a J. A. Holmes Safety Association certificate in 1931 for hacing operated from December 10, 1927, to December 10, 1930, or three years, without a lost-time accident with a produc- of 721,594 tons of ore in 802 days and 1,044,768 man-hours of mine operation.

5. Pickands, Mather & Company has six groups of iron-ore mines in Michigan and has a competition for an annual prize. Each employee of the winning mine in each group receives a prize. Any mine with a no lost-time record for the year is considered as tied for first place and is awarded the prize. On the Menominee Range there are two competitive groups, the Gogebic-Marquette district has three groups, and the Minnesota has two groups. In addition to the yearly competition individual awards are made for no-disability records at each mine. These awards are based on a cumulative credit system. When a mine works a month without disability, each employee receives a cash credit of 10 cents. If this is followed by a second month of no disability, 20 cents more is credited to each man. Underground, for a third and each consecutive month, 30 cents is credited. In the Minnesota open pits, due to better accident experience, the rate remains 20 cents for the second and subsequent months. These credits accumulate for each man as long as there are no-disability months. The total credits at the end of the months are as follows:

Months	Underground	Open pits
1	$0.10	$0.10
2	.30	.30
3	.60	.50
4	.90	.70
5	1.20	.90
6	1.50	1.10
7	1.80	1.30
8	2.10	1.50
9	2.40	1.70
10	2.70	1.90
11	3.00	2.10
12	3.30	2.20

When a lost-time accident occurs, a prize for the no-disability period is decided upon and each employee in the mine receives his award. After this the accumulation of credits starts over again.

The prizes are based on the amount of credits accumulated and are given in the form of merchandize as blankets, pocket knives, wool socks, fishing tackle, or similar materials. Inasmuch as the company can purchase wholesale, and gives the men the benefit of the wholesale price, the amount of the credit represents more in actual goods than the money would purchase at retail prices.

Partly as a result of this excellent bonus system, Pickands, Mather & Co. has a remarkable record in the safe operation of mining properties, and a treatise might well be written based upon its safety mechods and safety accomplishments. Employing approximately 3,000 persons at its 28 to 30 iron-ore mines in the Lake Superior Region, this company has done wonders in the long-time operatiom of mining plants without lost-time accidents. In 1927 out of 360 mine momths of operation (30 mines each 12 months) 192 or 53.3 per cent were without a lost-time accident; in 1928 there were 244 no-disability mine months, or 67.8 per cent out of a total of 360; in 1929 out of 372 mine months 266 or 71.5 per cent were no-disability; in 1930 there were 279 no-disability months out of 339, or 82.3 per cent; and in 1931 there were 275 (or ~~8.1~~ per cent) no-disability months out of 339. In other words, in both 1930 and 1931 every mine operated by this company worked an average of nearly 10 months of each year without the occurrence of a lost-time accident.

South Dakota

Copper, iron, and zinc mines are by no means the only metal producing organizations that have effectively functioning safety work with the bonus as an important part. At its great gold producer at Lead, S. Dak., the Homestake Mining Co. uses its own product (gold) to stimulate interest in safety and to reward not only mine officials but also other employees for good safety performance.

A bonus of $10 in gold is given every year to every underground worker with 300 or more shifts completed during the year without a lost-time accident, and any one earning this bonus for five successive years receives $20 in gold. At first sight it would appear to be an impossible performance for any person to work 300 shifts per year without considering the added difficulty of doing this without getting injured; however, it is stated that out of about 700 employed underground at this great mine 238 persons received the $10 bonus in 1923, 280 in 1924, 275 in 1925, 286 in 1926, 201 in 1927, 190 in 1928, 247 in 1929, and 360 in 1930. Moreover, for having earned the $10 bonus for five years in succession, 32 persons received the $20 bonus in 1927, 13 in 1928, 9 in 1929, and 12 in 1930.

To reward the bosses, a monthly bonus is given every underground foreman whose underground crew completes 300 or more shifts without a lost-time accident. Also frequently and severity bonuses are paid to bosses as follows (Oct. 12, 1931, issue of the Engineering and Mining Journal):

> A frequency bonus of $10 is paid annually and semiannually to foremen whose crews complete from 300 to 500 shifts per month with a frequency rate below 62.5 injuries per million hours; for 500 shifts or more the frequency bonus is $20. If the severity rate for the year is less than 0.5 shift lost per 1,000 hours worked, a severity bonus of $20 is also awarded. One fatal injury eliminates this reward for the shift foreman and the assistant foreman.

Tennessee

At Mascot, Tenn., and at Joplin, Mo., the American Zinc, Lead & Smelting Co. has mines; at Granby, Mo., and at East St. Louis and Hillsboro, Ill., it has smelters; and at Columbus, Ohio, it has a zinc oxide plant. Safety is well to the fore in importance with production and cost at these plants. At Mascot there is a general safety committee which makes quarterly inspections of the entire property. Monthly bonuses are paid to foremen for good safety records, based on the man-hour exposure of the men working under their direction. Every wage employee who works a year without a lost-time injury also receives a cash bonus. At the East St. Louis plant is also a general safety committee. Suggestions on safety are invited, and if any is adopted the man is credited.

As with other progressive mining companies which are doing good work in safeguarding the lives of their employees, the bonus is but part of an extensive accident-prevention system; but, as usual, it is an important cog in the wheel. Some idea of the excellent work being done at Mascot may be gained from the fact that between 1925 and 1929 there was a reduction of over 91 per cent in the occurrence of lost-time accidents.

Utah

The Utah copper Co. at its Magna Plant credits every employee with 10 cents for each month that the plant operates without a lost-time accident. this amounts to $60 per month at present, and the money is used to assist the needy and unemployed in and around the town of Magna. The company adopted this plan instead of giving its employees a smoker, a free show, or a dance.

CONCLUSION

The giving of some form of bonus for good safety performance in the mining and allied industries is becoming much more prevalent than in the past, even though some organizations which had safety bonus (or bonus and penalty) systems some years ago have abandoned them. It is significant that in many if not most of the cases where mining companies have made or are making outstanding

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records in safe operation of their properties, a safety bonus of some sort is in effect, at least as concerns the mine officials or some of them.

Much of the opposition to the bonus for safety performance is that not only each boss or mine official but, to even a greater extent, each individual owes it to his family, to his employer, and to himself to do everything in his power to prevent accidents. This unquestionably is true, but it is equally unquestionably true that every human being appreciates commendation; and in the last analysis the safety bonus, whether given in form of money, or as a certificate, or a flag, or a trophy, for some safety accomplishment is merely an expression of the giver's appreciation of good work on part of the receiver of the gift.

Whether the bonus is the $10 gold piece given by the Homestake Mining Co. for working 300 or more days per year without accident, or $5 per month to one, two, or three underground workers with the best-kept working places for the month, or a knife or a ham awarded to a worker in a mine or section of a mine which has operated for six months or a year without a lost-time accident, or possibly merely a safety flag flown at the mine or part of mine which has operated with the lowest accident frequency or severity rate - or, in fact, almost any other form of safety bonus, there is no question that when carefully planned and impartially administered the safety bonus tends more definitely to hold the attention of mine workers and bosses on prevention of accidents than almost any other one type of accident-prevention activity.

At times it seems difficult to secure the active cooperation of the workers in the prevention of accidents, and unquestionably the workers in some instances take a more or less "hard boiled" stand that it it the company's duty to prevent accidents and that the individual has little or no responsibility for preventing injuries to himself or his co-workers; almost invariably in such cases the entire situation is changed, almost overnight, by the announcement of some form of safety bonus in which each individual participates, particularly if that bonus gives the individual an increase in pay, even though it be small, for working considerable periods without lost-time accident or for some other type of good safety record of the individual or of the mine or section of the mine in which he works.

Mining companies have good reason to believe (even though they can not in many cases submit actual proof) that workers in some instances deliberately injure themselves - in other words, are malingerers; in several cases workers who are not overly fond of work or who are given but little work and can secure more remuneration from compensation than from working are known to have deliberately cut off part or all of a finger or a toe. This deplorable type of "accident" can be eliminated, or at least its performance made dangerous to the prospective malingerer by the installation of some form of safety bonus. For instance a worker who was slightly injured two days before the end of the month could have returned to work as far as his injury was concerned on the following day

but instead he "went on a drunk", costing his mine a lost-time accident and preventing his co-workers as well as himself from getting a ham for a no-accident month; this man was treated to so many sound beatings by his co-workers that it is improbable that he will again cause a lost-time accident unless it can not be avoided. The wideawake mining company can by instituting a well-thought-out safety-bonus system, so enlist the cooperation of the individual workers, including the bosses, that every individual will have a definite enough interest in the avoidance of accidents to consider it not only a personal loss but even a personal disgrace if he or any of his co-workers sustain an accident.

Data of a more or less fragmentary but at the same time very informative nature compiled during the past few years indicate that at least 10 per cent of the cost of mining our ore or coal is due to accidents; some progressive mining companies, recognizing this as a heavy and utterly unnecessary economic waste, have enlisted the active cooperation of the workers in reducing this drain by dividing on a half-and-half basis with the worker the amount saved by reducing the cost of accidents. If this or some similar type of safety bonus were made universal in mining, there is no question that within a very few years the occurrence of accidents would be as well under control in mining as it now is in the cement, railroad, and other industries which lead in safety and which until a very few years ago had accident rates little if any better than those of mining.

I. C. 6626 June, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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METHOD AND COST OF DREDGING SAND AND GRAVEL, PORTLAND GRAVEL CO., PORTLAND, OREGON

BY

HOWARD F. PUARIEA

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METHOD AND COST OF DREDGING SAND AND GRAVEL, PORTLAND GRAVEL CO., PORTLAND, OREG.[1]

By Howard F. Puariea[2]

INTRODUCTION

This paper is one of a series describing methods and costs of recovering sand and gravel from alluvial deposits in the United States and deals directly with the methods employed by the Portland Gravel Co. in operating a clamshell dredge on the Willamette River near Portland, Oreg., and a pump dredge on the Columbia River near Vancouver, Wash.

These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent local expenditures only and not total production costs. It is recognized that publication of total costs may in some instances cause embarrassment to individual producers as well as to the industry as a whole. On the other hand operating costs are essential to the technical discussion and study of methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

HISTORY

The Portland Gravel Co. was organized on March 20, 1925, and at once started construction of the clamshell dredge "Boulder" from designs by H. P. Warren and the writer. This dredge was put in operation on the Willamette River near Portland, Oreg., in July, 1925. Few alterations were necessary and the dredge has been in almost constant operation since that time. On May 1, 1926, the company purchased the pump dredge "Sandy" from the Columbia Sand Co. This dredge was designed and built in May, 1924, by the writer. With this equipment the Portland Gravel Co. had the largest capacity in the Portland district and during 1926, 1927, and 1928 furnished sand and gravel for all the major work in the vicinity and in addition supplied customers at Astoria, Oreg., and Longview, Wash. The purpose of the company is to wholesale material delivered on customer's barges alongside the company's dredges. No attempt has ever been made to enter the retail business.

GEOLOGY

The gravel used in the Portland district comes mostly from the bed of the Willamette River near the city of Portland. Outlying districts are supplied partly by material from dry pits and banks. The gravel is found in the river bars containing sand, gravel, silt, and débris. In the part of the river near Portland the bars shift but little due to the narrow channel and reefs of solid rock which protect the head of the bars. Many of these gravel bars contain water-soaked chips deposited in the river by the paper mills at Oregon City 12

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S Bureau of Mines Information Circular 6626."

2 - One of the consulting engineers, U. S. Bureau of Mines, and president of the Portland Gravel Co.

miles upstream. When these chips occur in quantity they ruin the deposits because they can not be separated economically; bars containing them are avoided by moving the dredge unless there is a market for fill material which can be loaded directly into barges and not run through the plant. In workable bars the upper gravel to a depth of 10 to 15 feet generally contains driftwood and débris but can usually be run through the plant without ruining the product. Beyond this depth the deposits are exceptionally clean.

The gravel ranges in size from pea gravel to bowlders 8 to 10 inches in diameter, and is mostly loose, although there are sections of hardpan and cemented gravel which are avoided if possible when dredging. The gravel is largely basalt and is hard and sharp. There are sections in the river that contain a high percentage of decomposed or rotten rock that when mixed with the good gravel spoils it for anything but fill.

The sand consists largely of rounded particles of basaltic material and rarely contains more than 10 or 15 per cent of quartz.

DREDGING CONDITIONS

Near Portland the river deposits have been reworked many times and prior to 1921 most of the dredges dumped the bowlders and excess sand back into the river. This refuse practically covers the bars and makes dredging difficult. Since 1921 Government engineers have required all dredges to crush the oversize or move it ashore. Excess sand is still wasted and causes considerable trouble.

The sand as taken from the river is dirty and contains up to 25 per cent of excess fines. In order to build up or properly grade this material it is necessary to waste these fines and to add the coarse material obtained from the crusher fines. The crusher makes about 25 per cent of fines below 3/8 inch so there is plenty of this material to add to the sand. The prepared sand weighs about 2,600 pounds per cubic yard and is so harsh that it requires much more work to put in place than Columbia River sand, although it is stronger.

When sand is saved, about 15 to 18 per cent of the material dug is wasted back to the river. When wasting sand, this proportion increases to about 40 per cent. On an average about 30 per cent of the finished gravel is crushed material.

The river is full of old snags and buried trees which would prove disastrous to a ladder dredge. With the clamshell dredge it is possible to dig around these obstructions. During the freshet period in winter, dredging is further complicated by swift water and drift which fills the excavation with trash and drifting waste sand.

CHOICE OF METHOD

A clamshell dredge was selected as being the cheapest means of recovering the gravel. The decision was based upon the volume needed, the conditions in the river bed, and the great depth of the deposits from which a suction or ladder dredge could not recover material economically.

PROSPECTING AND SAMPLING

Prospecting is done by moving the dredge from one bar to another. In some of the bars the material is fine and in others coarse. The different sizes of gravel are run through the dredge, finished, and samples taken and put through sieve tests. If necessary, changes are made in the screens to grade the gravel to meet specifications. The average weight of the gravel as prepared for market is 2,850 pounds per cubic yard. There has never been any

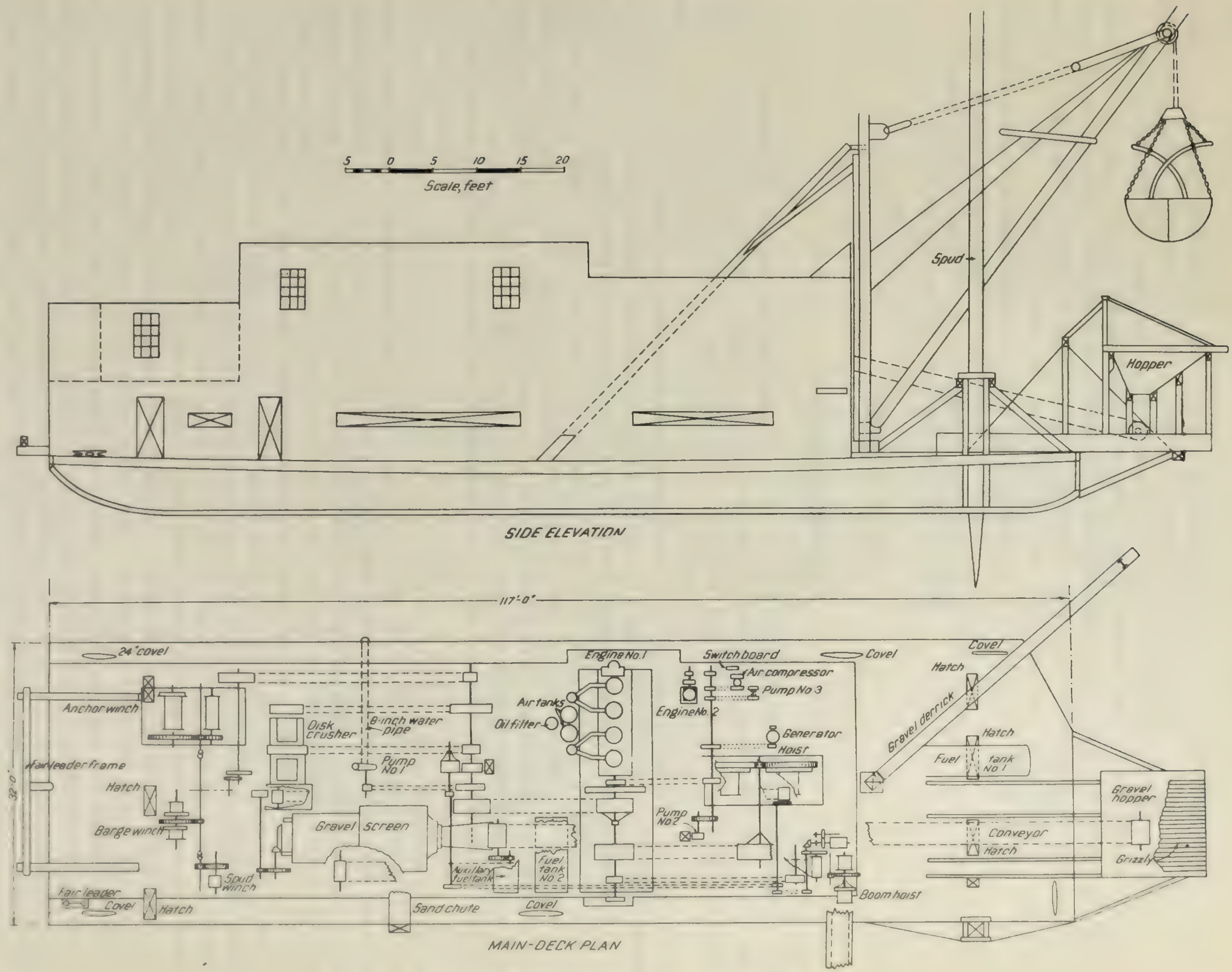

Figure 1 - Side elevation and main-deck plan of dredge "Boulder"

attempt to estimate accurately the reserve in these river bars as there is an unlimited amount of material which will last for years to come.

The prospecting and sampling of Columbia River sand is handled in the same way. A screen test is made of every barge load to determine the fineness modulus. The sand samples of both Willamette and Columbia River sand are taken from the barge with a sand augur. This augur is made from a piece of 1-inch pipe with slots cut in it at 6-inch intervals for a length of 5 feet. A fin is welded on one edge of each slot so that in turning the pipe it will take a sample at each fin. A sample is obtained in this manner at every few feet across the barge and is then tested with standard sieves.

Columbia River sand prepared for market will average 2,250 pounds per cubic yard. There is an unlimited amount of sand in this section of the river.

DREDGING GRAVEL

Dredge "Boulder"

The material is lifted by the clamshell to the overhanging hopper on the end of the dredge. (See fig. 1.) This clamshell bucket was built by the company from their own patterns and is made of cast steel with lips, pins, sheaves and all wearing parts of manganese steel.

The clamshell holds 40 cubic feet when level full. It is rigged with two 1-inch cables 225 feet long, one a closing line and the other a trip or holding line. The closing mechanism is the same as that of the well-known Williams 2-line clamshell bucket. An ordinary stiff-leg derrick is used. The boom is of wood and is 65 feet long with two 30-inch diameter sheaves in the top which lead the cables directly to the hoist through two 18-inch lead blocks swung from the A frame. This method of rigging works well, as the boom seldom travels more than a quarter turn. No turntable is used but pull lines from each side of the boom provide radial movement. The derrick is designed to lift 20 tons and the hoist for a straight pull of 10 tons.

The clamshell discharges into a 12 by 14 foot hopper built of wood. Three sides of the hopper slope and the fourth side is vertical and is fitted with a steel gate 14 inches square which is operated by hand. Material is fed through this gate into a cylindrical trunion-type trommel turning 30 r.p.m. and fitted with $\frac{1}{4}$-inch mesh wire cloth made of spring steel. This screen dewaters the gravel before discharging it on to the No. 1 conveyor belt. It also removes the sand from the gravel. No extra water is used in this operation.

The No. 1 conveyor belt is level, 30 inches wide, 75 feet between centers and has an average capacity of 200 tons per hour at a speed of 250 f.p.m. It discharges onto a washing table 5 feet long, 3 feet wide, and 18 inches deep extending into the main screen. This washing table sits level and is lined with $\frac{3}{4}$-inch steel plate bolted to the bottom and sides. The material is broken up and washed from this table into the trommel by a 5-inch stream of water delivering about 600 gallons per minute. In the trommel the material is washed again by a system of jets fed by a 3-inch pipe delivering about 200 gallons per minute against a head of approximately 60 feet. Valves placed at the washing table regulate the amount of water passing through the 5-inch main and the 3-inch jet system. Both are supplied by an 8-inch pump, belt-driven from the main countershaft, as shown in Figure 1. The surplus water from this pump fills the 3-inch fire main and also cools the main Diesel engine. The balance is returned to the river through the drip pans under the conveyor belts.

The main trommel has a standard frame supported on 30 by 5 inch trunion wheels at the charging end and the usual shaft at the discharge end. It is driven by a belt from the main-engine shaft through a bevel gear and pinion at 11 r.p.m. This screen has a slope of 22

inches in its length of 20 feet. It has two jackets. The inside section is 46 inches in diameter and its first 16 feet is of 3/8-inch plate with holes 1-5/8 inches square spaced diagonally across the section. The lower 4 feet is a punched plate with 3-inch round openings. The first jacket, 6 feet in diameter and 16 feet long, is made of 5/16-inch plate with 1-1/8-inch square holes. The second jacket is 8 feet in diameter and 8 feet long and is built in two sections of spring-steel wire with 3/8-inch openings. A screen of this design was used to save space.

The oversize bowlders pass directly from the screen to a 48-inch disk crusher set to crush to 1-inch size and discharging to the No. 1 belt conveyor over the No. 3 conveyor (see fig. 3).

The No. 3 conveyor belt which is 20 inches wide, 58 feet between centers, and travels 250 f.p.m. carries an average of 42 tons per hour. This method of mixing the crushed stone with the gravel makes a uniform and satisfactory product.

Out of 100 tons of material dug by the dredge, about 40 tons will pass the 3/8-inch screen and the dewatering screen. This fine material is usually wasted, but when making sand about 25 tons out of the 40 tons of fines is saved. The 60 tons of material larger than 3/8 inch is screened and crushed to the desired size. When making two sizes of gravel simultaneously, there will be about 25 tons of 3/8 to 1-1/8 inch material and about 35 tons of 1-1/8 to 3 inch gravel in the material larger than 3/8 inch.

The finished material is delivered by conveyor belts to barges alongside. It is possible to load a barge on each side simultaneously, and loading in this way is necessary for paving jobs where the material must be furnished in two sizes.

The No. 2 conveyor belt is 24 inches wide, 54 feet between centers, and travels 225 f.p.m. This belt carries an average of 70 tons of 3/8 to 2 inch finished material per hour to No. 2A conveyor which is a 24-inch belt on 28-foot centers, traveling 225 f.p.m. and discharging to the barges on one side.

The No. 4 conveyor belt is 20 inches wide, 48 feet between centers and travels 225 f.p.m. This belt carries an average of 42 tons of 3/8 to 1 inch finished material per hour directly from the main screen to the barges on the side of the dredge opposite that served by conveyor No. 2A.

The minus 1-1/8-inch plus 3/8-inch material can be delivered to either No. 2 or No. 4 conveyor by shifting a gate in the hopper just below the main screen. This is done in order to eliminate the shifting of barges from one side of the dredge to the other. Figure 3 shows a complete flow sheet of the dredge.

Directly under the sand jacket on the main screen there is a sand settling box of the usual V-shaped type with a gate in each side, one leading overboard and the other to a 16-inch screw conveyor 15 feet long which discharges the settled sand to the No. 4 conveyor and thence to the barge. The waste gate is adjustable to regulate the amount of fines going overboard. The only wash water used is that coming through the main screen. The natural sand is built up, as previously stated, with the fines from the crusher. This system results in a uniform and satisfactory product.

All belts are made of 28-ounce 5-ply duck and have 1/8-rubber cover and 1/16-inch rubber back. All conveyor rolls are ball-bearing and all conveyors are fitted with drip pans continually flushed with water to carry the spill back to the river. The conveyors are driven from the main engine through shafts and pulleys by rubber belts. The power is furnished by a 4-cylinder full Diesel engine, 14-inch bore and 17-inch stroke, developing 240-hp. at 257 r.p.m. and mounted on a foundation of steel beams and concrete. This power plant is serviced by a 10-hp. auxiliary engine driving the usual air compressor, oil and water pumps, and generator. The hoisting machinery, anchor winches, barge winches and clutches were all built locally from designs furnished by the company.

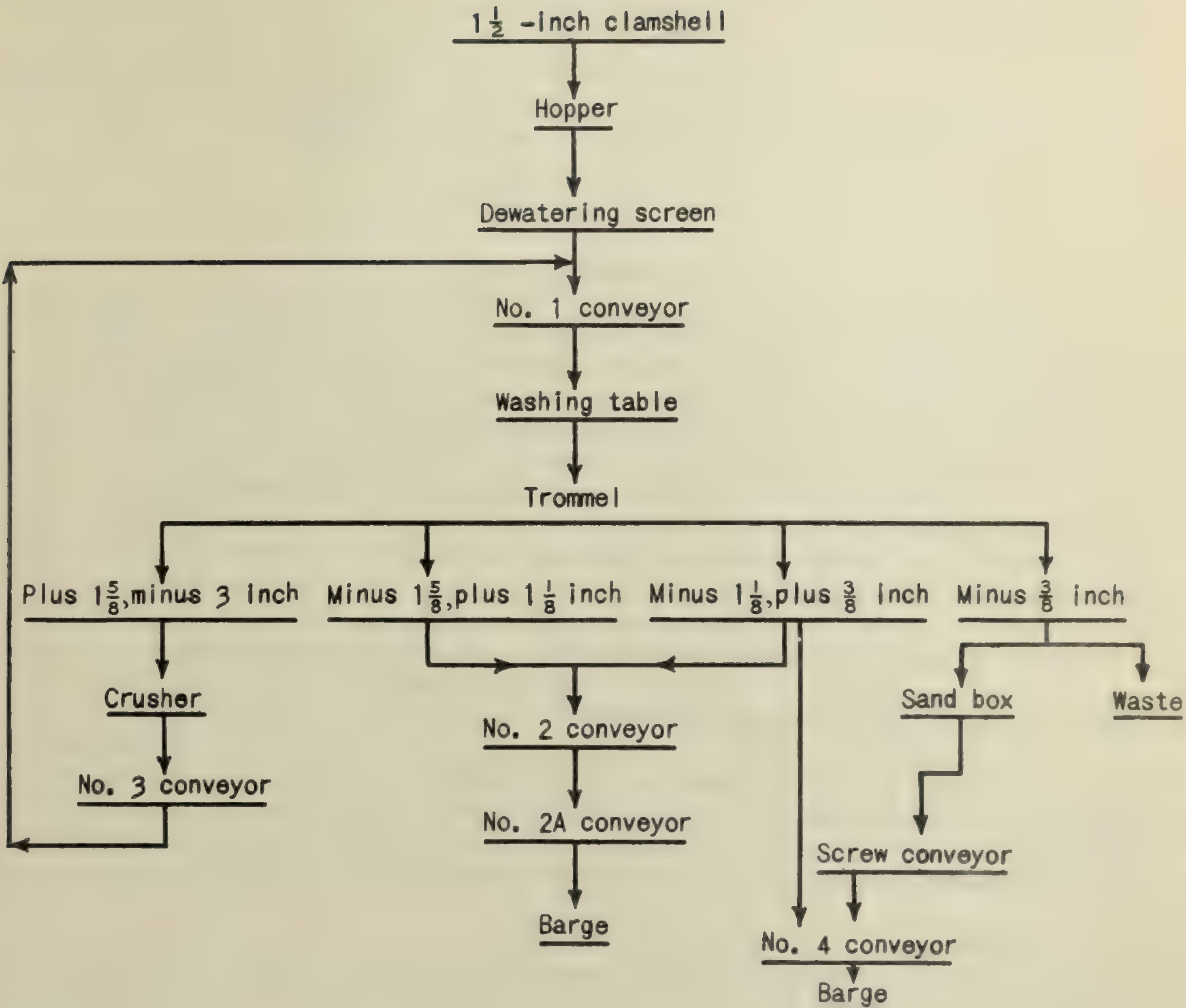

Figure 3.- Flow sheet of the dredge "Boulder"

The dredge hull is of wood and is 117 feet long, 32 feet wide and 6½ feet deep. No spuds are used but the dredge is held by four anchors, one at each corner to permit movement in any direction.

There is a 4-inch fire and barge pump that can be driven from either the main or auxiliary engine.

The average capacity of this dredge is 50 cubic yards per hour.

Living quarters for the crew are provided on the upper deck of the dredge.

The barges have a capacity of 300 to 500 cubic yards and are all built of wood.

DREDGING SAND

Dredge "Sandy"

The "Sandy" is a suction dredge with a wooden hull 90 feet long, 30 feet wide and 5 feet deep. The sand pump has a 10-inch suction and a 12-inch discharge. The impeller has three ports and is 28 inches in diameter. It is the closed type and is made of carbon steel. The pump housing is cast iron with a cast-iron liner on each side of the impeller. The main pump housing is replaced when worn, as this is cheaper than relining while dredging this kind of material. This pump makes 410 r.p.m. and operates against a head of 22 feet.

The main engine is a 75-hp., 2-cylinder semi-Diesel of 12-inch bore and 15-inch stroke, and makes 300 r.p.m. It is set on steel I beams which extend far enough to hold an outboard bearing which supports a tail shaft and a clutch pulley. There is another pulley on this shaft which drives the anchor machinery through a second clutch pulley.

A 3-hp. gas engine furnishes auxiliary power for the generator and air compressor. A 3-inch fire and barge pump is driven from the main engine.

The sand pump is driven by a belt from the main engine through a clutch to permit flushing out the pump.

There are two forward anchors on this dredge. No spuds are used as the current in the river holds the dredge in position. There are also three live drums attached to the anchor winches for moving barges alongside. These winches were built locally to fit the conditions and are driven from the main engine through clutches, as is also a small 2-drum winch that hoists the suction pipe.

The suction pipe line is made up of two 20-foot lengths of standard 10-inch pipe welded end to end and stiffened by three T irons welded at equally spaced intervals around this pipe and extending 10 feet along each section. A suction nozzle made of welded plate with a suitable screen over the end completes the suction line. This is connected to the main pump by a swivel joint and rubber suction sleeve to permit the pipe to swing vertically. The suction line hangs over one side of the hull on two davits. A cutter or agitator is not needed as the sand is loose. The pump handles from 18 to 20 per cent solids.

The sand pump discharges into a stationary cylindrical screen built up of three sections each having a different size of screen and covering one-third the circumference of the cylinder. The screen is mounted over a deep wooden flume. When in operation the screen is stationary and the bottom section is the only part in use. In changing sizes, the screen is turned so that the proper screen is at the bottom and the whole is then locked in position.

The oversize from this screen is rejected over the stern of the dredge. The sand drops through the screen into a deep flume, which can be raised or lowered, and is delivered to the barges alongside. About 20 per cent of the material pumped is wasted back into the river. The average capacity of this dredge is 175 cubic yards per hour.

To obtain different grades of sand it is only necessary to move the dredge to different sand bars.

Living quarters are provided on the dredge for the crew.

The barges loaded at this dredge all have sand boxes built on them fitted with sliding steel gates. These gates are at the bottom of the box at the deck level. The flume from the dredge extends across the barge and loading is started at one end and continued to the other, the water being forced through the gates by the sand. The gates are raised as the sand fills the boxes.

DIFFERENCE IN COST OF PRODUCING GRAVEL AND SAND

The cost of producing gravel is much higher than that of pumping sand, for the following reasons: The volume produced is considerably smaller, the gravel deposits are much more difficult to work, the maintenance on machinery is much higher, the cost of crushing is added, the power cost is higher, and the labor cost is twice as much.

The operation of pumping sand is simple and there are practically no interruptions, as the Columbia River deposits are clean and the percentage of waste material is fairly low.

The personnel and wage rates are as shown. All crews board themselves.

Personnel and wage rates

	Per day	Per month
Dredge "Boulder":		
Foreman		$175
Clamshell operator		175
Hopper man	$4	
Bargeman and watchman	5	
Dredge "Sandy":		
Foreman and operator		175
Bargeman and watchman		100

Table 1.- Cost of producing gravel

Dredge "Boulder" Period covered, January 1 to December 31, 1929.

Tons gravel produced, 84,000.

	Cost per ton
Power, including labor, repairs, fuel, oil, etc.	$0.0520
Clamshell derrick, including labor, repairs, cables, oil, etc.	.0292
Conveyors, including labor, repairs, oil, etc.	.0172
Screening, including labor, repairs, etc.	.0240
Curshing, including labor, repairs, oil, etc.	.0378
Miscellaneous operation, including salaries, superintendence, miscellaneous repairs, etc.	.0480
Royalty	.0600
Depreciation	.0690
Total operating cost	$0.3372

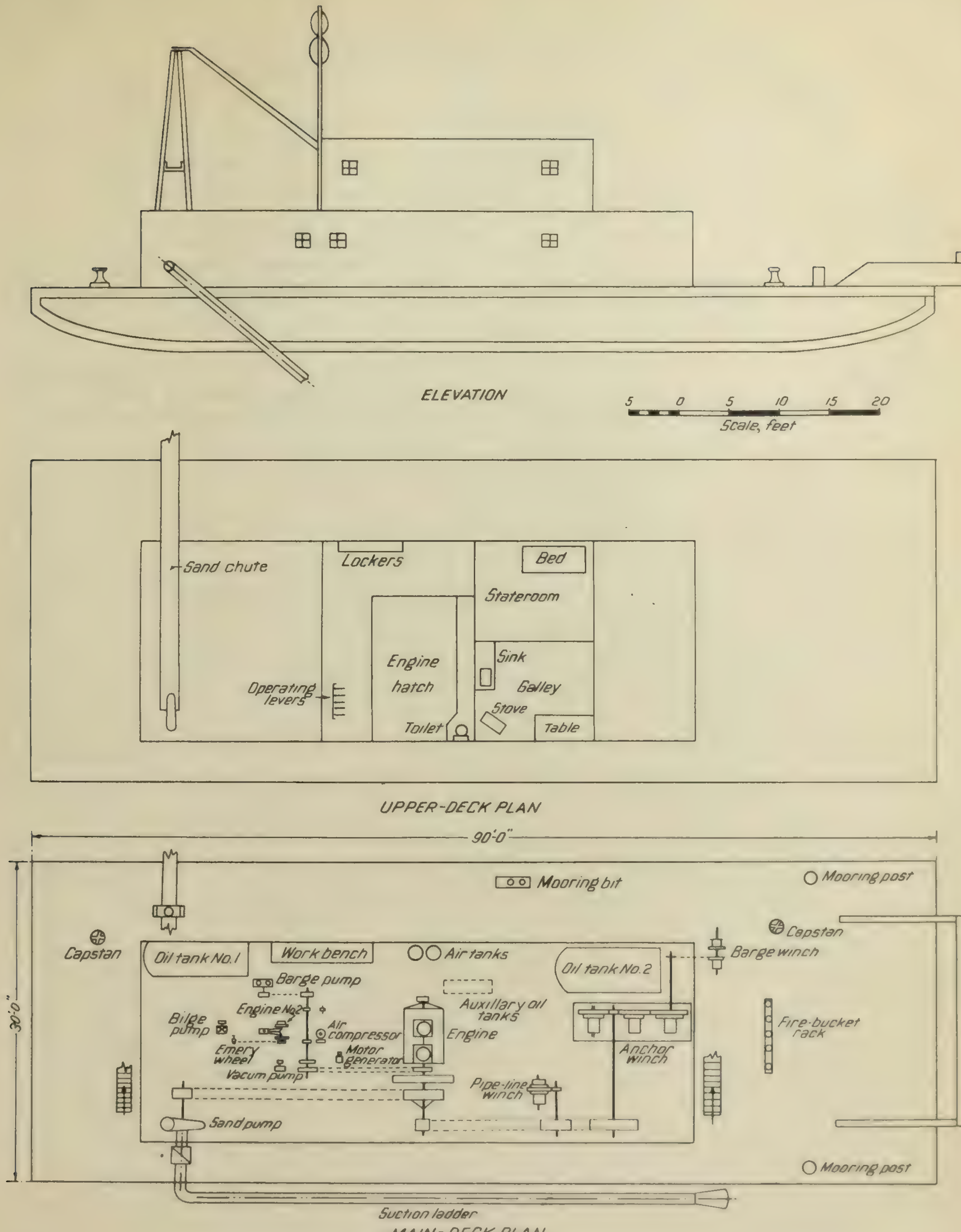

Figure 7.-Side elevation and deck plans of dredge "Sandy"

Table 2.- Cost of producing sand

Dredge "Sandy" Period covered, January 1 to December 31, 1929.

Tons produced, 126,500.

	Cost per ton
Power, including repairs, fuel, oil, etc.	$0.0080
Pumping, including pump repairs, parts, oil, etc.	.0132
Miscellaneous, including all labor, repairs, oils, cables, flume repairs, etc.	.0570
Royalty	.0437
Depreciation	.0220
Total operating cost	$0.1439

I. C. 6628

May, 1932

Department of Commerce

United States Bureau of Mines
Scott Turner, Director

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XXXVII

BY

Frederick W. Lee

INFORMATION CIRCULAR
DEPARTMENT OF COMMERCE - BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]
No. 37

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List of contributing editors of Geophysical Abstracts:

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1. GRAVITATIONAL METHODS

(749) LA PROSPECTION GRAVIMÉTRIQUE DU SOUS-SOL

(GRAVIMETRICAL PROSPECTING OF THE SUBSOIL)

By M. H. Galburn

Annales des Mines, Paris, vol. 20, No. 11, 1931, pp. 355-424.

In the first part of this article the author gives a mathematical derivation of formulas concerning gravity and the torsion balance, gradient of gravity, and the direction of the intensity of gravity at a point near the surface of the earth. Principles of gravimetrical prospecting, including correction for topography, are discussed.

The second part deals with the examination of the results of torsion balance measurements obtained at two stations (region of Gobian and Liege). Two maps showing the vector gradients and the isogams are added. Their interpretation forms the last part of the article.—W. Ayvazoglou.

(750) L'ANTICLINALE GRAVIMETRICO-PETROLIFERA DI FONTEVIVO

(GRAVIMETRICAL PETROLIFEROUS ANTICLINE AT FONTEVIVO)

By A. Belluigi

Extract from the Bolletin del Comitato Nazionale, Italiano per la Geodesia e la Geofisica. Second series, vol. 2, No. 2, Feb., 1932, pp. 1-3.

The note concerns gravimetrical exploration of the anticline in the region of Fontevivo, known for its petroliferous structure. Results of the investigation, accompanied by two maps, are given.—W. Ayvazoglou.

(751) THEORETICAL BASIS OF ISOSTASY

By W. D. Lambert

American Journal of Science, New Haven, vol. 21, No. 128, 1931, pp. 345-349.

The validity of observational proof of isostasy has been questioned on theoretical grounds by Hopfner, but his presentation of the mathematical theory is not generally accepted by geodesists. The author quotes Prey's views in support of the idea that the earth is isostatic. The Bruns term, the reduction from the geoid to the spheroid of reference of the gravity formula, can also be applied only qualitatively to explain an observed gravity anomaly. Criticism is also made of Hopfner's identification of the Bruns term with Bowie's reduction, but Heiskanan shows that the concepts are different.—R. S. Read. Abstract reprinted from Science Abstracts, vol. 34, No. 404, Aug., 1931, p. 665.

2. MAGNETIC METHODS

(752) TEST-DEFLECTIONS FOR VARIOMETERS AND MAGNETOGRAPHS

By George Hartnell

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No.4, 1931, pp. 279-296

This paper develops equations describing the deflections of a suspended magnet produced by a deflecting-magnet fixed in any desired position. The distances are assumed to be large enough to render distribution effects negligible. The equations are applied individually to a declination (D), horizontal intensity (H), and vertical intensity (Z) variometer, and to an assembly of these three variometers, which constitute a magnetograph. Under the conditions assumed, the equations furnish a convenient means of testing magnetic instruments. Since such instruments contain one or more magnets as essential parts, their response to magnetic forces as theoretically deduced provides a criterion of their merit.—Author's abstract.

(753) MAGNETIC SECULAR VARIATION FOR EPOCH 1930

By C. C. Ennis

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No. 4, 1931, pp. 315-317.

This paper is a contribution in the field of magnetical investigation to J. Bartels' article "Geophysical stereograms" published in vol. 36, No. 3, 1931, of Terrestrial Magnetism (Geophys. Abs. 33, p. 334). Ennis makes available the data employed in the construction of stereogram 8 which shows the secular variation of the magnetic field-vector at the intersections of the meridians 0°, 20°, 40°, etc., with the circles of latitude 60°N., 40°N., - - - 40°S.

As the resulting salient features of the secular variation were pointed out by Bartels, the author gives here only the data upon which the stereogram depends. They are shown in a table.--W. Ayvazoglou.

(754) LES ÉLÉMENTS DU MAGNÉTISME TERRESTRE À JASSY EN 1931

(THE ELEMENTS OF THE TERRESTRIAL MAGNETISM AT JASSY IN 1931)

By St. Procopiu

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No. 4, 1931, pp. 341-343.

In this article the author gives the results of measurements of magnetic elements carried out by him, together with Gh. Vasiliu, in 1931 near Jassy (Roumania), where a magnetic anomaly was established, and compares his data with those obtained by Hepites and Murat in 1898.—W. Ayvazoglou.

(755) PRELIMINARY REPORT OF THE MAGNETIC OBSERVATIONS MADE DURING THE AEROARCTIC EXPEDITION OF THE GRAF ZEPPELIN, 1931.

By Gustaf S. Ljungdahl

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No. 4, 1931, pp. 349-355.

The magnetic work during the arctic expedition of the airship Graf Zeppelin in July, 1931, was undertaken for the purpose of studying the possibilities of making magnetic measurements under the conditions involved and of obtaining as valuable observations as possible. A preliminary report of the results is presented in this article.

The determinations of the horizontal intensity (H) were made with the double compass. This investigation was subsidized by the Carnegie Institution of Washington. The results are given in a table.

An attempt was made to determine the declination (D) with a Thomson compass rose with fibre suspension, fashioned according to a model of Dr. Haussmann, by projection of the sun's shadow on the card. Only a few determinations were made, owing to the difficulties of taking the sun's bearing from the compass position.

The undamped compass rose often was subject to great and irregular oscillations and vibrations. Accordingly the results are not considered to be very exact.

For measurements of the vertical intensity (Z) Schmidt's field balance was mounted in gimbals and taken on board. The sensitivity was diminished from 30 γ to about 200 γ per scale division. No measurements could be obtained, owing to slipping of the magnet system caused by vibrations of the airship.--W. Ayvazoglou.

(756) GEOLOGISTS HEAR TALK ON GEOPHYSICAL METHODS

Editorial note

The Oil and Gas Journal, Tulsa, vol. 30, No. 37, 1932, p. 121.

In a lecture before the Shreveport Geological Society on January 21, 1932, W. M. Barret outlined the principles of the magnetic method of prospecting. Measuring instruments used in the field and laboratory apparatus developed for improving the technique of interpreting the field data in terms of geologic structure were discussed.--W. Ayvazoglou.

(757) ZU FOLGHERAITERS BESTIMMUNGEN DES MAGNETISCHEN ERDFELDES AUS DER MAGNETISIERUNG GEBRANNTER TONGEGENSTÄNDE

(CONCERNING FOLGHERAITER'S DETERMINATIONS OF THE MAGNETIC EARTH FIELD FROM THE MAGNETIZATION OF BAKED EARTHEN-WARE)

By. J. Koenigsberger

Gerlands Beiträge zur Geophysik, Leipzig, vol. 35, No. 1, 1932, pp. 51-54.

About 35 years ago Folgheraiter determined the magnetic inclination i at the Roman and Etrurian epochs by measuring the direction of the remanent magnetization in vases, assuming that they were baked standing on their bases. Folgheraiter found i to be about the same as at the present time ($+57^{\circ}$) in the year 100 after Christ and from $+2^{\circ}$ to $+25^{\circ}$ in the years 700 and 600 B.C.

Examination of the following questions was made by Koenigsberger to prove Folgheraiter's conclusions:

1. Is it possible to calculate exactly the inclination by measuring the three perpendicular components of the remanent magnetization?
2. Is the direction of the inclination influenced by quick changes of the temperature?
3. Does the additional burning of black color (as is used on antique vases) cause any change?
4. How great is the quotient $J_r : K \cdot T$ for similar materials, T being the present-time total intensity and K the susceptibility?

The experiments have shown that:

1. The accuracy was about $\pm 1^{\circ}$.
2. The remanent inclination was not changed by the temperature differences while baking.
3. The additional burning must be made at about 300° in order that no changes of the previous remananent magnetization could be caused.
4. Values similar to those of the most eruptive rocks were found.

The small values of inclination obtained by Folgherait for the Etrurian epoch can probably be explained by the supposition that the Etrurians baked the vases by putting them in horizontal position, or that they rebaked them in horizontal position while fastening the handles.—W. Ayvazoglou.

(758) INVERSION DE L' INCLINAISON MAGNÉTIQUE AUX ÂGES GÉOLOGIQUES. NOUVELLES OBSERVATIONS

(INVERSION OF MAGNETIC INCLINATION THROUGH GEOLOGICAL AGES. NEW OBSERVATIONS)

By P. L. Mercanton

Comptes Rendus, Académie des Sciences a Paris, vol. 192, No. 16, 1931, pp. 978-980.

In this article the author discusses facts collected from the study of the remanent magnetism of various samples of volcanic lava from both hemispheres (Greenland, Spitzbergen, Australia), indicating that at Tertiary epochs, at the time of large volcanic eruptions, the terrestrial magnetic inclination was opposite to that existing now. Magnetic inclinations of a series of samples are given.--W. Ayvazoglou.

(759) MAGNETISMUS RIPU (IN BOHEMIAN) (MAGNETISM NEAR RIP)

By V. Spacek and B. Zahálka

Rozpravy 11, Tridy Ceske Akademie, Prague, vol. 39, No. 48, 1929, pp. 1-15.

Theoretic and practical remarks on the sensitivity of Schmidt's variometer are given in the introduction to the article.

Results of the measurements of the differences

$$\Delta V = V - V_0$$

in which V is the vertical component of the terrestrial magnetism near Rip are given. The results are shown in a map.

The article is concluded by a supplement written by B. Zahálka in which the results of geophysical measurements carried out by Spacek are compared with the geological survey carried out by Zahálka.

According to the results it was established that there were two systems of faults in the tectonic of the surroundings of Rip, one W. NW. - E. SE. and the other S. SW. - N. NE. Rip is situated at the point of intersection of the two most important faults. These faults are characterized not only by magnetic anomalies but also by geological structure; thus there is no doubt about their existence. The results proved also the hypothesis established on this subject by Professor Laska. This hypothesis was the main reason for carrying out the measurements.--Authors' abstract translated by W. Ayvazoglou.

(760) STANOVENI AZIMUTU PRI PODROBNEM MERENI MAGNETICKE DEKLINACE (IN BOHEMIAN)

(DETERMINATION OF THE AZIMUTH IN THE ACCURATE MEASUREMENT OF MAGNETIC DECLINATION)

By V. Spacek

Special reprint from the Zememericsky Vestnik, No. 7, 1930.

In measuring declinations the astronomic determination of the geographical meridian usually takes much time. In this article the author shows that on a small area it is sufficient, in carrying out the measurements, to determine the direction of the meridian at one place only.

At the other places the measurement is made by angles only, as in Pothenot and Hansen problems.

By the method described the effect of the various geological formations on the magnetic inclination can be determined over a small area at as many places as at which can be measured the vertical component by Schmidt's variometer during one day--Author's abstract translated by W. Ayvazoglou.

3. SEISMIC METHODS

(761) TRAVEL-TIME CURVES AT SMALL DISTANCES AND WAVE VELOCITIES IN SOUTHERN CALIFORNIA

By B. Gutenberg

Gerlands Beiträge zur Geophysik, Leipzig, vol. 35, No. 1, 1932, pp. 6-45.

The study of local shocks in California may now be made with considerable accuracy in many cases, owing to the system of seismological stations in this country.

Gutenberg's studies of travel-time curves at small distances and wave velocities in southern California are described in this article under the following headings:

1. The seismological stations. A list of stations with data concerning them is given in Table 1. The location of most stations is shown in a figure.

2. The shocks. Only those shocks were used whose epicenters seemed to be certain within a very few kilometers. Data on the shocks used for the investigations are shown in Table 2. Table 3 shows calculation of epicenters and

origin times, and Table 4 contains an enumeration of seismograms for investigation of travel times.

3. The $\overline{P}$ waves. By using the epicenters given in Table 2 and the corresponding distances for every shock, the apparent velocity of the $\overline{P}$ waves was calculated. Table 5 shows the result. Table 6 gives the observed travel time of the $\overline{P}$ waves minus distance divided by 5.55. The principal result of Table 6 is that at distances between 44 and 86 kilometers the waves generally arrived 0.3 second earlier than would correspond to an apparent velocity of 5.55 kilometers per second. Between 100 and 200 kilometers this difference decreased to a mean of 0.1 second; in the interval which followed the difference was practically zero; at distances of over 400 kilometers the difference was nearly always negative, and the mean passed below -1.0 second between 500 and 600 kilometers. The most probable values of the travel times of $\overline{P}$ in southern California are given in Table 7.

4. P_n and P_x. The travel times (calculated from the observed times of arrival of P_n and the origin times as given in Table 2) are shown in Table 8. Travel times of P_n and difference in the travel times of $\overline{P}$ and P_n according to various investigations are shown in Table 9. In Table 10 the observed times of the P_x wave and the observed time differences $P_x - P_n$ are given. Table 11 shows the mean travel times of the P_x wave in California, the mean differences $P_x - P_n$ and the corresponding values found by Conrad with respect to the Schwadorf (Austria) shock.

5. Other waves between P_n and $\overline{P}$. Between P_n and $\overline{P}$ there are some other well-marked phases. They can be divided into two classes: Waves of the first class are well marked at short distances. The second class of waves between P_n and $\overline{P}$ is noticed especially at greater distances. In the seismograms of the California shocks two kinds of waves of the first class could be found -- P_m and P_x -- and three kinds of the second class -- a, b, and c. The time differences of all these waves compared with P_n are plotted in a figure.

6. S waves and surface waves. Observed travel times and mean travel times of S waves and surface waves are given in tables.

7. Other waves. It is known that if a longitudinal or a transversal wave touches a surface between two different layers, it divides into four waves. Since there are several surfaces of this kind within the upper 40 kilometers of the earth's shell, a large number of reflected and refracted waves must be expected. Probable travel time curves of such waves at various distances are given in figures.

8. The depth of foci. From the results of the investigation the author derived that very probably all shocks originated at a depth between 10 and 15 kilometers.

9. The thickness of layers in southern California. Assuming the depth of a focus at 12 kilometers, the thickness of the first (granitic) layer is calculated to be 14 kilometers. This thickness of 14 kilometers agrees very well with the value of 15 kilometers found by Byerley (Bulletin of the National Research Council, No. 61, p. 86, Washington, July, 1927), Wood and Richter (Bulletin of the Seismological Society of America, vol. 21, p. 28, 1931) from blasts.

References complete the article.--W. Ayvazoglou.

(762) MIT WELCHER GENAUIGKEIT LÄSST SICH DIE SCHALLGESCHWINDIGKEIT IN DER STRATOSPHÄRE FINDEN?

(AT WHAT ACCURACY CAN THE VELOCITY OF SOUND IN THE STRATOSPHERE BE DETERMINED?)

By B. Gutenberg

Gerlands Beiträge zur Geophysik, Leipzig, vol. 35, No. 1, 1932, pp. 46-50.

In a recent paper (Velocity of Sound and the Temperature in the Stratosphere; Geophys. Abs. 21, p. 28), Gutenberg tried to show that in general the height at which the temperature of the stratosphere begins to increase more rapidly is greater in summer than in winter. There are very many effects due to unavoidable errors, especially the effect of the wind at heights above 10 kilometers, in general unknown. A simple method has been used by Gutenberg to calculate the results within the limits caused by the errors. Some objections to this method were raised by F. J. W. Whipple in his article entitled. On Methods of Estimating the Heights Reached by the Air-Waves Which Descend in Zones of Abnormal Audibility, published in Gerlands Beiträge, Vol. 31, No. 1-3, 1931, pp. 158-163.

In this article Gutenberg examines the objections raised by Whipple.--A. Ayvazoglou.

(763) ON THE RAYLEIGH WAVE PROPAGATING OVER THE SURFACE OF A HETEROGENEOUS MATERIAL

By H. Honda

Geophysical Magazine, Tokyo, vol. 4, No. 2, 1931, pp. 137-145.

The problem of the surface wave of Rayleigh's type propagating along the surface of a heterogeneous material, of which Lamé's constants λ and μ increase

linearly with increasing depth and the density remains constant throughout the material, was solved approximately under some assumptions. And the expressions showing the velocity, the dispersion of the wave propagation, and the displacement components of the particles of the material and so on were obtained.--Author's abstract.

(764) REPORT ON THE ACTIVITY OF THE EARTHQUAKE RESEARCH INSTITUTE, TOKYO IMPERIAL UNIVERSITY, DURING THE SECOND HALF OF 1930

By Chuji Tsuboi

Gerlands Beiträge zur Geophysik, Leipzig, vol. 35, No. 1, 1932, pp. 113-122.

This is the fourth report in which Tsuboi summarizes the results of an investigation carried out by the Earthquake Research Institute during the second half of 1930 (for previous reports see Geophys. Abs. 10, 16, and 24).

The items of this report are as follows:

1. Seismic activities in Idu Peninsula. An outline of the scheme of investigations carried out by the members of the institute is given. The results will be published in due course.

2. Instrumental. M. Ishimoto elaborated an accelerometer for the purpose of observing earthquake accelerations. With this instrument it is possible to obtain records of earthquake accelerations on smoked paper solely by mechanical means. The proper oscillation period of the pendulum of the accelerometer is 0.15 second and the statical magnification 200.

3. (1) Terada has calculated the heat generated in a deformation of the earth's crust. (2) Terada, Miyabe, and Tsuboi have calculated the areal expansions and contractions of various earthquakes and volcanic districts by using as data relative displacements of triangulation points.

4. Statistical. (1) Yamaguchi studied the aftershocks of the Kwanto earthquake of 1923, of the Tango earthquake of 1927, the Tazima earthquake of 1925, and of the Oomachi earthquake of 1918 with respect to tidal heights in the seas nearest to the respective seismic region. (2) Takayama and Suzuki made a statistical study of the relation between sunspot activity and the occurrence of destructive earthquakes in Japan.

5. Mathematical. Sezawa and Nishimura discussed mathematically the deformation of a single shock with its propagation through an elastic medium for two cases; one was related to the propagation of a shock through a medium composed of different parts of different elastic constants and densities, with

their boundaries parallel to the wave front, while the other was related to the propagation of a shock through a medium which is horizontal and dispersive in itself.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(765) PROGRESS AND PROBLEMS OF ELECTRICAL PROSPECTING (IN RUSSIAN)

By Parfenov, Melikian and Nikitin

Azerbaidjanskoe Neftianoe Khoziaystvo, Baku, vol. 12, No. 1, 1932, pp. 7-14.

Schlumberger's method for studying the formations penetrated by drill holes, "electrical coring," and its application in the oil region of Azneft in the U. S. S. R. are discussed. The development of this method during the year 1930-1931 is illustrated by a figure. A series of diagrams showing the results of investigations of drill holes by the electrical coring method is given.

The authors conclude that this method is of great importance in geological interpretation of the subsoil and should be used on a large scale.--W. Ayvazoglou.

(766) GOLD DEPOSITS OF NORTH SWEDEN

Editorial note

The Mining Journal, London, vol. 176, No. 5035, 1932, pp. 124-125.

The important possibilities of the Boliden district in north Sweden, the main deposit of which was discovered in 1924 by electrical methods of prospecting, are described. Of the many indications obtained by the geophysical method, only comparatively few of the more promising indications were tested by the drilling.--W. Ayvazoglou.

(767) DER EINFLUSS DER ANISOTROPIE DER GESTEINSMEDIEN AUF DIE VERTEILUNG NIEDERPERIODISCHER, ELEKTROMAGNETISCHER WECHSELFELDER

(INFLUENCE OF THE ANISOTROPY OF THE EARTH'S ROCK MEDIA ON THE DISTRIBUTION OF LOW-PERIOD ELECTROMAGNETIC ALTERNATING FIELDS

By Max Müller

Gerlands Beiträge zur Geophysik, Leipzig, vol. 30, No. 1/2, 1931, pp. 142-195.

Contents of the paper: (1) Introduction and the objects of the research. (2) Apparatus for the production and measurement of small period electromagnetic alternating fields of any low frequency. (3) Calculation of the current field of an electric dipole in connection with frequency, conductivity, and thickness of strata. Setting forth of the conditions for the disappearance of the source-

free secondary current. (4) The influence of the anisotropy of the rock media on the distribution of stationary currents. (5) Graphic calculation of the magnetic field of a double source with consideration of the decrease of the current density with depth. (6) Electromagnetic field measurements by means of a current directly conducted to the earth: (a) Experimental research on the influence of frequency and anisotropy on the distribution of the components of the magnetic field of a double source; exposition of a new method for the determination of the mean conductivity of the earth; (b) investigation of a series of strata by means of low-frequency alternating current; (c) measurement of a deposit of Thuringian magnetite by means of low-frequency alternating current; (d) investigation of a barite-pyrite deposited by means of low-frequency alternating current. (7) Theory of the inductive methods: (a) Calculation of the magnetic field of a stationary circular current; (b) investigation of the dependence of frequency of eddy currents induced in the earth's crust by a closed alternating current; (c) conclusions drawn from the theories of Levi-Civita and Pollaczek. (8) Inductive field measurements by means of alternating currents of various frequency: (a) Investigation of the dependence on frequency of the distribution of the magnetic field of a right-angled cable traversed by an alternating current; discussion of the influence of anisotropy on the Jena shell limestone; (b) investigation of a series of strata by means of low-period electromagnetic alternating fields; (c) investigation of the magnetic external field of a low-period circular current upon anisotropic rock media.

The paper is illustrated by 51 figures including diagrams of apparatus and curves showing the results of observations.--W. Ayvazoglou.

(768) GEOLOGICAL CORRELATION OF CROSS-SECTIONS ACCORDING TO INFORMATION OBTAINED FROM LENIN'S REGION (IN RUSSIAN)

By B. Sarkisianz

Azerbaidjanskoe Neftianoe Khoziaystvo, Baku, vol. 12, No. 1, 1932, pp. 14-21.

Investigations of drill holes in the Lenin's oil-bearing region by means of Schlumberger's "electrical coring" are described. A series of cross sections of the holes is given. According to the author, more tests are required to draw definite conclusions on the degree of usefulness with which electrical coring can be applied in the region described.--W. Ayvazoglou.

(769) RESULTS OF THE APPLICATION OF THE "ELECTRICAL CORING" IN STALIN'S REGION (IN RUSSIAN)

By M. Babaian, V. Listengarten, V. Gorin and G. Zaturov

Azerbaidjanskoe Neftianoe Khoziaystvo, Baku, vol. 12, No. 1, 1932, pp. 21-27.

Electrical coring was applied in Stalin's oil-bearing region for the first time in January, 1931. One hundred and eighty drill holes were investigated during one year. In this article the authors examine the results obtained from the application of Schlumbergerger's method by comparing the diagrams of the cross sections of a series of drill holes.

Summing up the results of the investigation the authors say that electrical coring must be recognized as one of the most useful and necessary methods on determining the correlations in the stratigraphical conditions of oil deposits.--W. Ayvazoglou.

(770) RESULTS OF THE APPLICATION OF THE ELECTRICAL CORING IN THE ORDJONIKIDZE'S REGION (IN RUSSIAN)

By D. Jabrev and M. Emelianov

Azerbaidjanskoe Neftianoe Khoziaystvo, Baku, Vol. 12, No. 1, 1932, pp. 28-30.

Two hundred and twenty drill holes were investigated from October, 1930, to January, 1932, by Schlumberger's electrical coring method.

The problem was to prove, based on the results from the interpretation of the data obtained, that stratigraphical conditions of the ground can be determined in most cases by electrical coring only, and that thus the expensive and slow procedure of mechanical coring can be avoided. On a series of diagrams the solution of this problem is examined and the great advantages of electrical coring are positively established.--W. Ayvazoglou.

(771) THE FIRST PRELIMINARY SUMMARIES OF ELECTRICAL CORING IN KIROV'S REGION (IN RUSSIAN)

By S. Movsesian

Azerbaidjanskoe Neftianoe Khoziaystvo, Baku, vol. 12, No.1, 1932, pp. 31-33.

Electrical coring has been used in Kirov's region since November, 1930, as it was proved that the correlation of the stratigraphical conditions of the subsoil could be determined and the oil-bearing and water-bearing horizons established by this method. Almost all the holes drilled since March, 1931, were investigated by electrical coring. Whether the mechanical coring can

entirely be substituted by this method or not can not, according to the author, be definitely decided at present.

A series of diagrams showing the cross sections of the holes and a plan of Kirov's region are added.—W. Ayvazoglou.

5. RADIOACTIVE METHODS

(772) FURTHER EXPERIMENTS ON THE UNIFORMITY OF DISTRIBUTION OF THE COSMIC RADIATION

By Robert A. Millikan

The Physical Review, Minneapolis, vol. 39, No. 3, 1932, pp. 391-396.

More careful and prolonged observations on the small, daily variations before reported in the measured intensities of the cosmic rays, the new observations being made under such conditions as to eliminate the possibility of a slight temperature effect suggested by Bowen and Millikan's recent explanation of ionization-pressure relations in high-pressure electroscopes, yield the definite result that "within the limits of the author's present observational uncertainty, which is of the order of a third of a per cent, the sun has no direct influence on cosmic-ray intensities." New evidence is presented that if observed and apparently systematic variations of the order of a third of a per cent are in fact real, they are best interpreted as the result of small changes in the blanketing effect of the earth's atmosphere due to air currents.—Author's abstract.

(773) INVESTIGATIONS OF KENNELLY-HEAVISIDE LAYER HEIGHTS FOR FREQUENCIES BETWEEN 1,600 AND 8,650 KILOCYCLES PER SECOND

By T. R. Gilliland, G. W. Kenrick, and K. A. Norton

Bureau of Standards Journal of Research, Washington, D. C., vol. 7, No. 6, 1931, pp. 1083-1104

The contents of this paper are as follows:

I. Introduction.
II. 4,045-kilocycle observations.
III. Virtual layer heights as a function of frequency during daytime.
IV. Diurnal variations.
V. Analytical discussion of results: (1) Refraction; (2) reflection phenomena.
VI. Conclusions.

The results of observations of the height of the Kennelly-Heaviside layer carried out near Washington, D. C., during 1930 are presented. Evidence for the

existence of two layers (corresponding closely in virtual height to the E and F regions discussed by Professor Appleton) is found during daylight on frequencies between 3 and 5 megacycles. The modification in the virtual height of the higher F layer produced by the existence of a low E layer is investigated theoretically, and the possibility of large changes in virtual height near the highest frequency returned by the E layer is pointed out.

A number of oscillograms showing the characteristic types of records observed during the tests are presented, with a graph of average heights for January to October, 1930.--Authors' abstract.

(774) SYSTEMATIC ERRORS IN MEASUREMENTS OF IONIC CONTENT AND THE CONDUCTIVITY OF THE AIR

By O. H. Gish

Gerlands Beiträge zur Geophysik, Leipzig, vol. 35, No. 1, 1932, pp. 1-5.

It is pointed out that the possible source of error in ionic content and air-conductivity measurements made with the charging method, which were discussed in recent papers by Yo Itiwara (Geophys. Abs. 25) and by J. Scholz (Geophys. Abs. 2[illegible]), were recognized when this method was adopted in the Department of Terrestrial Magnetism of the Carnegie Institution of Washington, and that suitable provisions were made to obviate such errors. The design of ion counter which embodies these provisions is described in detail.

The effect of large ions upon the ionic-content measurements, and of distortions of the earth's field in the region from which air is drawn for both ionic-content and air-conductivity measurements, are discussed. It is concluded that these sources of error are negligible for the conditions prevailing on the "Carnegie" at sea.—Author's abstract.

(775) ACTION OF THE EARTH'S MAGNETIC FIELD ON PENETRATING RADIATION

By B. Rossi

R. Accademia Nazionale dei Lincei, vol. 13, January 4, 1931, pp. 47-52.

Referring to his experiments in which he confirmed Bothe and Kolhörster's results, which showed that even at sea level there exists a penetrating ultra-radiation, the author describes further researches made to test whether the penetrating radiation (cosmic radiation) is affected by the magnetic field of the earth. If the penetrating radiation reaches the limit of the atmosphere as a corpuscular radiation having an energy of some thousands of volt-electrons, the terrestrial magnetic field should give place to a strong dissymmetry in its distribution of intensity with respect to the plane of the magnetic meridian. Experiments carried out to put in evidence this dissymmetry gave a negative result. --J. J. Stewart; Reprinted from Science Abstracts, vol. 34, No. 407, 1931, p. 9[illegible].

(776) ÜBER EXAKTE INTENSITÄTSMESSUNGEN DER HESSCHEN ULTRASTRAHLUNG

(EXACT MEASUREMENTS OF THE INTENSITY OF HESS' COSMIC RADIATION)

By G. Hoffmann

Zeitschrift für Physik, Berlin, vol. 69, No. 11/12, 1931, pp. 703-718.

For obtaining higher accuracy in measuring the intensity of cosmic radiation according to the ionization method, not only a great amount of gas but also a full compensation of charges is necessary. The results of experiments carried out with the large Twin-apparatus (Zwillingsapparat) are shown by curves. A strong meteorological influence on the soft radiation is apparent. In the case of hard radiation there is noticeable, next to the dependence on pressure of the air, also a direct or indirect influence of the sun. The problem of a proof, free from objection, of the possible existence of a sidereal time period seems to be a very difficult one.--Author's abstract translated by W. Ayvazoglou.

(777) ÜBER NEUE ARBEITEN AUF DEM GEBIETE DER KOSMISCHEN ULTRASTRAHLUNG

(CONCERNING NEW WORKS ON COSMIC RADIATION)

By V. F. Hess

Elektrotechnische Zeitschrift, Berlin, vol. 52, No. 29, 1931, pp. 936-937.

The purpose of this article is to inform the readers on several new important works concerning ultra-radiation which appeared since Regener's lecture delivered at the meeting of the Electrotechnical Association and Heinrich-Hertz Society in Berlin (November 18, 1930). This article deals especially with the ionization of the higher layers of the atmosphere and with the latest views of the nature and origin of the cosmic ultra-radiation.--Author's abstract translated by W. Ayvazoglou.

(778) RÉPARTITION ANGULAIRE DES RAYONS ULTRAPÉNÉTRANTS (RAYONS COSMIQUES)

(ANGULAR DISTRIBUTION OF COSMIC RADIATION)

By D. Skobelzyn

Comptes Rendus de l'Académie des Sciences, Paris, vol. 194, No. 1, 1932, pp. 118-121.

The author shows the distribution of cosmic radiation between the various angular zones as determined from statistical material collected by him.

The results are given in two figures.--W. Ayvazoglou.

(779) SUR LA THÉORIE DE L'AURORE POLAIRE

(ON THE THEORY OF THE AURORA BOREALIS)

By A. Dauvillier

Comptes Rendus de l'Académie des Sciences, Paris, vol. 194, No. 2, 1932, pp. 192-194.

The purpose of this article consists of verifying the theory concerning the aurora borealis, based on new observations made at the Sodankylä magnetic station (Lapland). The following conclusions drawn by the author are given briefly:

1. The observations confirm entirely the opinion that the aurora represents the secondary effect of the initial cosmic phenomenon which occurred very far away from the earth.

2. The phenomenon consists of two phases: The first corresponds to the initial cosmic effect and sometimes lasts only a very short time; the second is phosphorescence, losing slowly its brightness, due to the excitation, ionization, and polymerization (ozone) produced by secondary electrons.

3. The phenomenon presents a simple theoretical aspect only when the intensity remains weak. As soon as it increases the phenomenon becomes more complicated by secondary effects which are no longer of the cosmic nature.

4. Photometric estimation of the brilliancy of a screen observed in the zenith made it possible to calculate the energy spent during an aurora storm by admitting the same luminous yield (10^{-2}) as that in the case of a vacuum tube.

5. These observations were accompanied by measurements of the gradient of the atmospheric potential.

6. The measurement of the intensity of cosmic radiation showed fluctuations of 2.5 per cent, or five times greater than the errors of the measurement; they could not be attributed to variations of atmospheric pressure or to the magnetic activity.—W. Ayvazoglou.

(780) RADIOAKTIVITA DISLOKACI NA PRIBRAMSKU (IN BOHEMIAN)

(RADIOACTIVITY OF THE DISLOCATIONS IN THE DISTRICT OF PRIBRAM)

By Jaroslav Splichal

Rozpravy 11, Tridy České Akademie, Pregue, vol. 40, No. 24, 1930, pp. 1-20.

The district of Pribram offers the occasion for measuring the radioactivity of soil-air not only in the sediment rocks but also in the granite

and in the contact. The radioactivity was evaluated by pumping soil-air out of a bore hole 25 to 35 centimeters deep into the ionization chamber placed on a Wulf's electrometer.

The results of the measurements were as follows:

According to Profile I, shown in the appendix, the activity of soil-air in the granite (490 volts per hour) was much greater than that in the Algondian slates (104 volts per hour).

The radioactivity of the soil-air was the greatest in the granite; average results were 2.2 to 3.6 Mache or 6.10^{-10} grams of the element radium per liter of the soil-air measured.

Algondian schists had an activity of 0.9 Mache or 1.5×10^{-10} grams of the element radium.

The size of the emanation in the Cambrian conglomerates and Cambrian graywackes depended on the amount of detritus of igneous rocks.

The greatest activity was found in the conglomerate of Zitec ($C\alpha_1$) 1.2 Mache or 2.10^{-10} grams of the element radium.

The conglomerate of Hluboš ($C\alpha_2$) and the graywacke of Sadec-Bohutin ($C\alpha_3$) had the values of 0.2 Mache or 0.4×10^{-10} grams of the element radium.

The results of the measurements of radioactivity on the dislocations are graphically reproduced in the profiles. There the distances of the boreholes are plotted as abscissas, the radioactive values of the soil-air in volts per second as ordinates. The following relations are deduced from the radioactive profiles:

1. The fault between the granite and the schist has a fluctuating radioactivity of the soil-air, the value of which is the <u>mean</u> of that of the granite and of the schist.

2. The dislocations in the schists show a maximum activity.

3. The activity on the dislocation in the granite between the two complexes is smaller than in the granite itself.

The measurements of radioactivity on the "clay-split" had also shown that the radioactivity is greater in the dislocation than in the neighborhood.--Author's abstract.

6. GEOTHERMAL METHODS

(781) TEMPERATURE MEASUREMENTS IN THE DEEP CIECHOCINEK WELL (IN POLISH)

By Jan Moniak and Stanislaw Zych

Kosmos, Lwow, vol. 55, No. 1-2, 1930, pp. 423-427.

The results of temperature measurements carried out on July 28, 1929, in the deep well Ciechocinek No. 14 are given in the following table:

Time during which the thermometers were left in the well		Temperature of the outside air, °Cel.	Depth, meters	Separate readings of the thermometers			Temperature accepted, °Cel.
Hours	Minutes						
2	25	19	50	31.5	31.5	31.5	31.5
1	40	18	100	31.55	31.55	31.55	31.55
2	25	19	200	31.6	31.55	31.55	31.55
1	40	18	300	31.7	31.65	31.65	31.65
1	40	18	400	31.95	31.9	31.9	31.9
1	40	18	500	32.15	32.05	32.05	32.05
1	40	18	600	32.2	32.1	32.1	32.1
1	40	18	700	32.45	32.45	32.4	32.45
1	40	18	800	32.7	32.7	32.7	32.7
2	25	19	900	33.05	33.0	32.95	33.0
2	25	19	1,000	33.5	33.5	33.5	33.5
2	25	19	1,100	34.0	33.9	33.9	33.9
2	25	19	1,140	34.0	34.0	34.05	34.0

The purpose of the measurements was to establish the temperature of thermal salt waters required for baths.--W. Ayvazoglou.

7. UNCLASSIFIED METHODS

(782) ABSTRACT OF THE INNSBRUCK MEETING OF THE COMMISSION OF TERRESTRIAL MAGNETISM AND ATMOSPHERIC ELECTRICITY OF THE INTERNATIONAL METEOROLOGICAL ORGANIZATION AND OF THE RESOLUTIONS ADOPTED SEPTEMBER 21-23, 1931

By H. D. Harradon

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No. 4, 1931, pp. 319-323.

In this article Harradon gives a summary of resolutions adopted by the Commission of Terrestrial Magnetism and Atmospheric Electricity of the International Meteorological Organization on the following principal items:

1. Publication of the magnetic character-numbers.

2. Relations between the commission and the Association of Terrestrial Magnetism and Electricity of the International Union of Geodesy and Geophysics.

3. The various questions pertaining to the work during the Polar year.

4. Study of the relations of the moon and the magnetic elements.

5. The question of the location of new observatories for terrestrial magnetism and electricity.--W. Ayvazoglou.

(783) ABSTRACT OF THE INNSBRUCK MEETING OF THE INTERNATIONAL COMMISSION FOR THE POLAR YEAR 1932-33 OF THE INTERNATIONAL METEOROLOGICAL ORGANIZATION AND OF THE RESOLUTIONS ADOPTED SEPTEMBER 23-26, 1931.

By H. D. Harradon

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No. 4, 1931, pp. 324-332.

The principal matters brought before the sessions of the commission were indicated in the report of President D. la Cour, an abstract of which is given by Harradon.

In the second part of the article Harradon gives a summary of resolutions adopted at the Innsbruck meeting.--W. Ayvazoglou.

(784) SUMMARY OF THE YEAR'S WORK, DEPARTMENT OF TERRESTRIAL MAGNETISM, CARNEGIE INSTITUTION OF WASHINGTON

By J. A. Fleming

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No. 4, 1931, pp. 333-340.

This is an extract from the annual report in Year Book No. 30 of the Carnegie Institution of Washington for the year July 1, 1930, to June 30, 1931.

The following principal items of the work carried out by the institution are mentioned: (1) Participation in formulation of plans for the Jubilee International Polar Year proposed in 1932-33; (2) interpretation of accumulated data and the study of magnetic correlations with other geophysical, solar, and cosmic phenomena; (3) study of magnetic records and tabulations derived from observations at Watheroo, Western Australia, during the 12 years 1919-1930; (4) investigation of correlation of the initial impulses of magnetic storms; (5) investigations in atmospheric electricity; (6) carrying out of programs at the Watheroo and Huancayo observatories concerning magnetic, atmospheric-

electric, earth-current, and meteorological observations; (7) demonstration by measurements of the artificially produced beta and gamma rays of energies equivalent to most of those emitted by radioactive substances, using equipment developed at the Department; (8) reductions and compilations of the work in physical and chemical oceanography from observations made aboard the "Carnegie"; (9) instrumental advances for magnetic and electric determinations.

The following theoretical investigations, mostly in continuation of those noted in the last year's summary, are given: (1) Solar activity and secular variation; (2) magnetic activity; (3) magnetic storms; (4) arctic magnetic charts; (5) magnetic work at sea and dynamic-deviation investigation; (6) photographic method of changing the ratio of ordinate scale to abscissa scale; (7) atmospheric pollution.

The first of a contemplated series of manuscripts giving detailed descriptions of the department's specially designed apparatus required in its research fields was completed by Torreson. A second manuscript by Gish on the earth-resistivity apparatus is in preparation.--W. Ayvazoglou.

(785) WISSENSCHAFTLICHE UND PRAKTISCHE AUFGABEN DER ANGEWANDTEN GEOPHYSIK

(SCIENTIFIC AND PRACTICAL PROBLEMS OF APPLIED GEOPHYSICS)

By O. Meisser

Berichte Freiberger Geologischen Gesellschaft, vol. 13, 1931, p. 44.

The translation of the abstract published by M. Henglein in the "Neues Jahrbuch für Mineralogie, Geologie, und Paläontologie," 1931, No. 5, p. 628, reads as follows:

The purpose of applied geophysics consists in carrying out structure investigations of physical fields on the surface of the earth. The pure physical problems consist of development the necessary methods by using corresponding instruments, and of collecting the proper constant data. Based on these methods practical application of geophysics in mining is possible; a great number of geological problems may be disclosed by corresponding geophysical surveys over large areas.

Improvements in instruments used for pendulum measurements are discussed. By using the three new Jena methods of observation and the apparatus for the relative gravity measurements the gravity difference can be determined simultaneously at several stations with an accuracy corresponding to that of the torsion-balance measurements.

The principle of the Eötvös torsion balance is explained by an example relating to brown-coal investigation. Combination of the torsion balance and pendulum is mentioned.

Concerning magnetic measurements some arrangements of H and Z components by which the variations of the components can be determined were shown. The progress in electrical methods of measurement is explained. The possibility of determining single layers by seismic methods is mentioned.

Attention is drawn to radioactive and geothermal measurements. Emphasis is laid upon the necessity of further developing instruments and methods. The importance of detailed measurements of smaller areas with regard to gravity and magnetic elements, and in solving geological and industrial questions is especially stressed.--W. Ayvazoglou.

(786) SUMMARY OF THE GEOLOGICAL CONFERENCE (IN RUSSIAN)

By A. M.

Azerbaidjanskoe Neftianoe Khoziaystvo, Baku, vol. 11, No. 11/12, 1931, pp. 3-7.

In this summary the author mentions Schlumberger's electrical coring and its usefulness in determining the character of formations traversed in drilling. Four French parties have been working in Baku since 1930.

Geophysical section reported on electrical, gravitational, and magnetic methods of prospecting carried out by the Azneft. The conference considered it necessary to have six parties for carrying out electrical coring in Baku and one each in the Grozneft, Turkmenneft, and Uzbekneft. The organization of a series of parties for electrical and magnetic prospecting was also decided.--W. Ayvazoglou.

(787) NEW OIL RESERVES (IN RUSSIAN)

By K. Riabinin and G. Helkvist

Azerbaidjanskoe Neftianoe Khoziaystvo, Baku, vol. 11, No. 11/12, 1931, pp. 101-105.

The authors describe a series of new probable locations of oil discovered by various geophysical methods of prospection. The most important of them are as follows:

Peninsula of Apsheron. Anticline structure was detected by electrical methods of prospecting at several places (Giurgiani-Zyria, Zykh, Kala, and others.

Prikurinsk Region (region along the Kura River). Several anticlines were discovered by electrical and magnetic methods of prospecting.

A series of geophysical exploration works is planned in many other regions. --W. Ayvazoglou.

(788) ELECTRIC MICROMETER APPLIED TO MEASURING VIBRATION AND STRAIN

Editorial note

The Iron Age, New York, vol. 129, No. 10, 1932, p. 620.

A vibration detector recently developed in the general engineering laboratories of the General Electric Co., Schenectady, is described. Somewhat like the seismograph for recording earthquake tremors, this vibration-detecting device is an application of the electric micrometer. The description of the development of a pressure detector and a strain gage employing the same principle is given also.

Some details of the devices and of their operation are discussed.

Photographs are added.--W. Ayvazoglou.

(789) GEOPHYSICS PAPERS RICH IN THEORY AND PRACTICAL DATA

By Sherwin F. Kelly

Mining and Metallurgy, New York, vol. 13, No. 303, 1932, pp. 118-119.

This is a brief report on articles presented at the sessions devoted to geophysics during the February meeting of the American Institute of Mining and Metallurgical Engineers in New York. Papers concerning various methods of geophysical exploration, as well as those of special interest to oil mon are enumerated.--W. Ayvazoglou.

9. NEW BOOKS.

(790) Stutzer, O. Erdöl. Allgemeine Erdölgeologie und Überblick uber die Geologie der Erdölfelder Europas (Petroleum. General petroleum geology and a short account of the geology of European oil fields). Gebrüder Borntrãger, Berlin, 1931, 628 pp., 199 figs., price $15; special to A.A.P.G. members $11.25. The book is essentially a compilation of the literature on the subject.--W. Ayvazoglou.

(791) Texas Gulf Coast Oil Scouts Association and South Louisiana Oil Scouts Association. Oil and Sulphur Development in the Texas and Louisiana Gulf Coast Salt Dome Region. Bulletin 1, Houston, 1931, 128 pp., maps, sections, tables. Contains compilation of data on the Gulf Coast.

(792) Woolnough, W. G. Report on tour of inspection of the oil fields of the United States of America and Argentina and oil prospects in Australia. 1931, 119 pp., 36 illus. Department of Home Affairs, Canberra, F. C. T., Australia. Price, 5 s. The book gives a brief account of the American oil geology and exploration

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3 - The first figure refers to the number of the abstract, the second to the method of prospecting as indicated in the Table of Contents, and the third to the page.

I. C 6629
May, 1932

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING LAWS OF POLAND[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the mining laws of Poland has been prepared almost entirely from the answers made by Joseph Flack, American chargé d'affaires, Warsaw, to a questionnaire instituted by the Bureau of Mines and transmitted through the courtesy of the Department of State. This digest is released subject to correction and amplification, if necessary, by American foreign-service officers.

INTRODUCTION

A new mining law for the Republic of Poland was promulgated by presidential decree dated November 29, 1930, published in the Journal of Laws (Dziennik Ustaw) No. 85, text 654, December 5, 1930, and was to become effective January 1, 1932, in all Poland except Silesia. Its adoption in Silesia was dependent upon the action of the Silesian Diet. As the new law is considered an improvement over the old Austrian Mining Law, which is operative in the Silesian districts of Bielsk and Cieszyn, and the Prussian Mining Law, which is operative in Upper Silesia, no delay in its adoption for Silesia was looked for.

This new law codifies the various juridical regulations pertaining to mines existing in different parts of the Polish Republic but does not include legislation with respect to the extraction of mineral oil, especially crude oil. Petroleum legislation was postponed to a later date in order to facilitate codification in other fields. Moreover, as the oil industry exists in Galicia alone, delay will not be detrimental.[3]

1 The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6629."

2 Rare metals and nonmetals division, U. S. Bureau of Mines.

3 Redfern, Gilbert, New Polish Mining Law: Messager Polonais, Warsaw, Jan. 5 and 7, 1931 (translation): Rept. 265, Warsaw, Jan. 14, 1931, Bur. For. Dom. Com. file 195351.

A presidential decree was promulgated December 3, 1930 (Journal of Laws No. 86, text 667, December 6, 1930), for the regulation of the coal industry for a period of three years. (See section of this paper entitled "Special Coal Legislation.")

RIGHTS OF FOREIGNERS

In order to carry on mining operations in Poland, foreigners (private persons or corporations) must receive authorization from the Polish Government. Permits for the purchase of mining properties are issued to foreigners by the Minister of Industry and Commerce (who has wide discretionary power), in conjunction with the Minister for Foreign Affairs. (Art. 16.) Prospecting rights are granted in a similar manner. (Art. 5.) A foreigner in possession of a permit for prospecting does not need a further permit in order to exercise mining rights if the results of his search are favorable. (Art. 16.)

Foreigners enjoy equal rights with natives whenever mining authorities recognize the necessity of the occupation of land for exploitation purposes. If a foreigner inherits by law and not by a special will, he does not require a permit in order to enter into possession of his inherited mining property.

A Polish joint stock company, even if the shares are owned by foreigners, need not obtain a permit to explore unless the deposits are in the frontiers or Government-controlled areas (see section of this paper entitled "Prospecting Permits"), restrictions being placed upon foreigners and natives alike with respect to certain territories.

Regulations relating to foreign mine managers are included in the section entitled "Miscellaneous."

OWNERSHIP

Under the new law, surface and subsoil rights are separate, as the principal idea of the law is that of property exclusive of subsoil wealth. The transition from the old law, under which no distinction was made between surface and subsoil ownership, has been made easy by the provision that during a determined period the owner of the surface shall have the exclusive right to request a mining concession for minerals under the surface of his land. So-called "mining exceptions," which in Cieszyn, Silesia, and in Galicia gave an exclusive right to explore within a radius of 425 meters from a test well, will remain in force for a period of 15 years, unless they are liquidated earlier through special regulations. It is evident that owners will prefer mining rights obtained under the new mining law, which fixes the area of a concession according to the "depth of the mineral layer" (250 hectares when the depth is 500 meters, 500 hectares when the depth is 500 to 750 meters, and 800 hectares when the minerals lie deeper), to those obtained under the old (Austrian) law, although the old law grants rights to all minerals found and the new law to one mineral only. Under the old Austrian law, amended by a Polish law of March 3, 1924, the owner of a

"mining exception" could obtain a maximum area of 96 hectares.

The new law reserves to the State the property rights to certain minerals that it has by virtue of existing laws: namely, in Upper Silesia and Posnania, by the amendment of 1909 to the Prussian law; in Pomerania, by the same amendment and by a presidential decree of March 22, 1923; and in former Austrian Poland, by the same decree and by the Austrian decree of March 11, 1835 (concerning customs duties and monopolies).

The reservation by the State of certain minerals (see the succeeding section of this paper) is defended for the following reasons: A Polish salt monopoly is already in existence; concessions for potassium salts to private persons would be tantamount to leasing them to foreign syndicates; and the reservation of coal in Upper Silesia is but the maintenance of the old Prussian law.[4]

CLASSIFICATION OF MINERALS[5]

The new law places minerals in but two main classes: Those reserved to the State and those that may be privately owned and exploited.

The State reserves for its exclusive property deposits of rock salt, sodium or potassium salts, and manganese, as well as deposits of pit coal and anthracite located in the Province of Poznan and in the upper part of the Province of Silesia, where the Government has the exclusive right to grant mining property (for pit coal and anthracite), provided it does not violate private rights granted prior to 1909.

Mining properties containing natural deposits of minerals, such as radium, gold, platinum, copper, tin, zinc, cadmium, lead, mercury, iron, cobalt, nickel, arsenic, antimony, manganese, aluminum, chromium, and wolfram, may be "privately owned and exploited for technical purposes and for mining the various minerals, under the existing mining regulations."

In addition, deposits containing minerals that can be used for the production of sulphur and for the manufacture of artificial fertilizers (phosphates) may be "privately owned and operated under existing regulations," as well as deposits of hard and soft coal, graphite, and anthracite not located in the Province of Poznan and in the upper part of the Province of Silesia.

EXEMPTED AREAS

Article 5 of the new law designates certain territories in which neither the Treasury of the Polish State nor Polish citizens nor foreign citizens are

4 See footnote 3.

5 Survey of Poland, The New Mining Law: Vol. 6, No. 288, Am. Polish Chamber of Com. and Ind. in the U. S. (Inc.), 149 East 67th St., New York City, Dec. 27, 1930, p. 877.

permitted to conduct mining operations, such as public squares, railways, highways, cemeteries, districts under mining protection, and sources of mineral waters.

The owner of the surface may refuse the right of prospecting within the precincts of buildings and at a distance of 65 meters therefrom or within gardens, orchards, parks, and courtyards.

PROSPECTING PERMITS

General.- In addition to the provision that foreigners must, in order to prospect, obtain authorization from the Minister of Industry and Commerce (in conjunction with the Ministers of War and Foreign Affairs), both foreigners and natives are subject to the following stipulations:

1. In the frontier area a permit must be obtained from the district administrative authorities (permits being issued from the standpoint of safety).

2. In territories administered by or for the use of military authorities or of State enterprises controlled by the Minister of War (for instance, ammunition factories) or in the precincts of fortresses, a permit must be obtained from the Minister of War.

3. Within the precincts of aerodromes, a permit must be obtained from the Minister of War and the Minister of Communications. (Art. 6.)

Consent must likewise be obtained from the owner of the ground for the right to explore. If the owner of the land refuses permission, the prospector may apply to the district mining office for a provisional permit and may, if necessary, appeal from his decision to the Mining Administrative Tribunal for a compulsory right to use the land for which a lease has been refused. Further dispute must be settled by the civil courts. (Art. 7, 84-94, and 97-98.)

The owner of the surface may not protest against subterranean works unconnected with the surface, and he is entitled to compensation for surface damages only. Remuneration for the use of the ground shall be double the usual rental value.

A prospecting permit is not an exclusive right, as free competition exists, except with respect to the "mining exceptions," for rights granted prior to January 1, 1932 (referred to in the section of this paper entitled "Ownership").

Prospecting area.- The size of a prospecting area is not specifically limited by the law.

Minerals to be sought.- The minerals that may be searched for under a permit include all minerals not reserved to the State.

Duration and renewal.- No limitation is placed upon the duration of an exploring permit except when the owner of the surface has objected, in which case the district mining office issues a permit for one year and may extend the period, provided work was not done during the year or was interrupted for more than one year. (Art. 7.)

Fees.- The new law provides for no fee for an exploring permit; the Austrian law provides for a special fee of 16 zlotys[6] for each exclusive right ("mining exception"). (Law of July 8, 1924, Dz. U. No. 69, test 67.) Nonpayment of this fee renders a permit void.

MINING

Mining property is obtained by virtue of a "bestowal," which the Government must grant whenever the applicant has fulfilled the legal requirements. (Art. 22 and 28.)

Application.- Applications for mining rights are made to the Chief Mining Office through the district mining office. (Art. 23 and 25.)

An application should contain: (1) Name and surname, or name of firm, and address (legal residence) of the applicant, that is, of the person requesting the mining rights; (2) citizenship of the applicant; (3) definition of minerals that will be covered by the permit; (4) accurate description of the place where discovery was made (by means of the boring point, adit, shaft, etc.); (5) the date of the discovery, in case the applicant desires to profit in priority rights (art. 37, clause 2); (6) the name to be given to the mining field; and (7) the address of the person that filed the application, together with evidence of his right to act as a representative, in case the application is filed on behalf of a person other than the applicant.

Failure of an application to fulfill stipulations 1, 3, and 4 renders it void. Clauses 2, 6, and 7, if lacking in the application, must be answered within two weeks from the date of summons from the mining authorities, or the application will be annulled.

Plans.- Within six months from the filing of the application, the applicant shall submit to the mining office of the district in which discovery is made a description of the future mining field, its dimensions in square meters, and a plan (in quadruplicate and on the scale prescribed by the Minister of Commerce and Industry), which shall show the point of discovery, objects on the surface that may serve for orientation, and the meridian. No extension, even on appeal, shall be allowed for the filing of the plan itself. The two months' period for filing omissions may be extended once for two months. Survey registers or cadester extracts of the area must be submitted (in quadruplicate) if mortgage regulations so require.

The plan shall be made by a qualified mining surveyor or by a sworn authorized surveyor. (Art. 163.)

6 Average annual rate of exchange of the zloty in 1930 was 11.205 cents.

Priority of right.- The prior right to a bestowal rests with privileged prospectors, that is; (1) A mining prospector, (2) the owner of a mining field that has discovered minerals as a result of his own operations, and (3) the owner of the surface that has worked it at his own expense (for instance by sinking a shaft). (Art. 37.) Preference shall be given to the prospector first reporting the results of his search to the district mining office. If he shall have filed his application within 14 days from the date of his discovery, the date of prior right shall be counted from the day of discovery.

Size of area.- An area of 250 hectares will be granted in case discovery was made at a depth of 500 meters; 500 hectares for a depth of 500 to 750 meters; and 800 hectares for a depth greater than 750 meters. (Art. 32.)

Duration.- No definite limit is placed upon the duration of a mining right; unless cancelled by the Government, it terminates only at the option of the holder.

Cancellation.- Cancellation of a bestowal may take place upon one condition only, namely, failure to exploit. If the High Mining Office (considering the exploitation of the mineral to be indispensable to the public interest) decrees that the mineral shall be mined, and if the owner fails to comply with the decree within six months from its issuance, cancellation of the mining right follows. The holder of the right may appeal, either when the obligatory decree is issued or when the mining right is cancelled, to the Mining Administrative Court (art. 173) and further to the Highest Administrative Tribunal (art. 222).

TRANSFERS

Mining property may be sold or rented under the general stipulations of the civil law. (Art. 20.) The administrative authorities must be informed of the transfer (art. 118), but only with respect to the responsibility of the "person running the mining enterprise for his own account for compliance with technical and police instructions."

MINING COLLEGE (BOARD OF APPEAL)

The mining law provides for a "Mining College," or "Administrative Board of Appeal," composed of representatives of chambers of commerce and communal unions. This body determines all cases of appeal, of first instance, from the decisions of authorities concerning mining concessions, the division of mining fields, the loss of rights, and like matters. An important function is that of resolving expropriation questions, as a second instance, after the district mining offices.[7]

RENT OF ROYALTIES

The assessment for mining rights is 4 zlotys annually for 1 hectare of mining property in the case of coal, 1 zloty in the case of iron, and 2 zlotys in

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7 See footnote 3.

the case of other minerals. Payment of this royalty is made in half-year installments. (Law of July 8, 1924, concerning fees for mining rights, Journal of Laws of Poland No. 69, text 671, and a modification thereof in 1928, Journal of Laws of Poland No. 21, text 180.)

SPECIAL COAL LEGISLATION

The presidential decree of December 3, 1930 (previously referred to), which was to be in effect for three years, vests the Polish Minister of Industry and Commerce with extraodrinary powers for controlling Poland's coal industry (the term "coal" meaning anthracite, brown coal, coke, and coal briquets). The practical import of the decree was to bring about a coal cartel (convention) among producers. The Government does not interfere with the repartition of coal between the members of the cartel but will help them to organize exportation.

MISCELLANEOUS

Qualifications of mine managers.- In principle, the managers of mining enterprises must be mining engineers with three years of experience in mines. Where no danger exists, however, the Minister of Industry and Commerce may admit to the management of a mine a person with a secondary mining education, provided he has worked six years in a mine. Polish citizens occupying managers' posts at the time the new law becomes effective may retain their positions on condition that they become registered within a period of six months.

As the new law (art. 3) states that the provisions thereof can not interfere with international agreements, regulation concerning personnel will not be applied to the German personnel in Upper Silesia until the Geneva convention shall have expired.

If the works' manager is not a citizen of Poland, he must obtain a permit from the Minister of Commerce and Industry. (Art. 133.) This provision relates to technical managers, who are responsible to mining authorities, but does not apply to economic managers.

Mine fillings.- The new law confers upon the owner of a mining field the right to demand permission from the surface owner to draw sand and other filling materials, in return for adequate remuneration. Formerly mine operators were often compelled to bring this material from distances, at great inconvenience. The increase in population in the vicinity of the mines has made it necessary to provide against subsidence of mining land.